STUDY OF STRATA BEHAVIOUR IN BLASTING GALLERY PANEL IN COAL MINES

A THESIS SUBMITTED IN PARTIAL FULFILLMENT OF THE REQUIREMENTS FOR THE DEGREE OF

BACHELOR OF TECHNOLOGY

IN

MINING ENGINEERING

BY

B.N.V. SIVA PRASAD

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DEPARTMENT OF MINING ENGINEERING

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CERTIFICATE

This is to certify that the thesis entitled, "***Study of Strata Behaviour in Blasting Gallery Panel in Coal Mines***" submitted for the award of the degree of Bachelor of Technology (Mining Engineering) in National Institute of Technology (NIT) Rourkela, is a record of original research work carried out by Sri B.N.V. Siva Prasad under our supervision. The context of this thesis has not been submitted elsewhere for the award of any degree to the best of my knowledge.

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My special thanks to the authority of National Institute of Technology, Rourkela for rendering permission to use their most modern and updated library in connection with the literature survey of the Project work.

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CONTENTS

ABSTRACT

In mining industry, the challenging task of a mining professional comprises of the extraction of maximum natural resources with utmost safety of the miners. This task becomes more problematic when the thickness of coal seam is larger. "Blasting Gallery" method is a unique technique of depillaring thick seams for higher recovery of coal. The extensive literature survey clearly helps to understand that the ultimate potentiality of the method is yet to be explored. Though a number of researchers, academicians and other stake holders attempted to work on it but impact of many significant parameters are still to be analyzed.

The Blasting Gallery operation in a mechanized underground mine system depends upon many decisions influenced by the geo-technical parameters which are often interspersed with inherent strata configurations. The present study has been aimed to examine thoroughly BG method operational systems in Indian geo-mining conditions such as:

- Study of roof convergence with respect to face advancement during different stages of extraction of coal in Blasting Gallery panels in SCCL mines.
- Simulation of field conditions in the numerical model generated using FLAC.
- Interpretation of strata behaviour through numerical modeling using FLAC.

In order to study the strata behaviour of such coal mines with thick seams, GDK 10 incline, 3A panel of Singareni Collieries Company Limited (SCCL), Ramagundam was selected. This mine has a thick coal seam of 11m and is at depth of 350mtr, practicing Blasting Gallery method to the maximum extent. Convergence behaviour with respect to goaf edge distance (GED) was monitored with the help of high state-of-the-art equipment (calibrated) throughout the life of BG panel. An over emphasis was given on the field study where data of BG mine specially related to natural falls, induce blasting etc. were recorded. Convergence of roof strata in mm, corresponding goaf edge distance (GED) in meter (m), corresponding distance from face in meter (m) and depth of panel in meter (m) were measured to know the significant impact of different layers with varying overburden pressure which leads us to think some logical sequence of interrelated operations.

The coal sample was collected from the mine and was tested for determination of the rock mass parameters. The geo-technical conditions of the mine were simulated and Numerical Modelling was carried out by using the most sophisticated software – FLAC. The output results obtained from the mine data was compared with that of model data and distance from goaf edge was considered as a sensitive variable so that the validation would represent the system in totality. The different conclusions drawn from this work is enumerated as follows:

- The maximum rate of convergence and cumulative convergence recorded in field was about 4mm/day and 61mm respectively, measured at convergence station C-5 in 68 Level.
- From the triaxial testing, the major principal stresses of 22, 32 and 41.5 MPa were obtained at confining stresses of 0, 2 and 3 MPa respectively.
- The results obtained from the RocLab software indicated the Cohesion, Friction Angle, UCS and Tensile Strength values to be 1.1MPa, 30.84^0, 1.314 MPa and 0.32MPa respectively.
- The model predicted maximum cumulative convergence to be 70mm while that observed in field is 61mm.
- The results obtained by FLAC when compared with that of the Field data, the predicted value were within an approximation of 10% for stages I, III, IV & V whereas for stages VI and VII, its in 20% approximation except for that of Stage II which showed a higher value of cumulative convergence measurements due to occurrence of natural fall.

LIST OF TABLES

LIST OF FIGURES

CHAPTER 1

INTRODUCTION

1. INTRODUCTION

Strata control or roof control implies the control of the strata to facilitate mining operation to be done efficiently and safely. Not only we are concerned with the roof falls and uncontrolled failure of strata or structure in the rock but also with the harnessing of the strata pressure to advantage so that there is ease in coal getting. There is less emission of gas and less production of dust and also the caved strata fills the goaf solid so that the risk of spontaneous heating is minimized. Obviously, the strata on the face, and in the adjoining area, i.e. in front and behind, must require attention so that no uncontrolled failure of the ground takes place. In order to design satisfactory strata control measures it is essential first to have a clear understanding of the mechanics of the movement of the ground as a result of mining operation.

In thick coal seams, coal bed forms the roof of the lower slices. A coal layer at the roof normally forms a good roof. But coals with joints and cleats are prone to fail without warning. Some seams have coal balls, nodules or rounded fragments, and these may fall unnoticed and cause fatalities. When coal bed is undermined, it may also develop induced cleavages and fractures and in such situations roof falls are common. So for complete or maximum extraction of such coal seams, the Blasting Gallery method is introduced.

First Blasting Gallery method of extraction was introduced in SCCL in 1989 at GDK No.10 Incline and being worked successfully. Although, first BG in India was introduced in East Katras Colliery of Jharia Coal Fields, BCCL and Chora Colliery of Raniganj Coal Fields, ECL in 1987, the workings were abandoned in East Katras Colliery due to Strata Control Problem, and were discontinued in Chora Colliery due to premature Spontaneous heating problem. GDK-10 Incline mine falls in Godavari Valley Coal Fields of Singareni Collieries Company Limited and is situated in Andhra Pradesh.

This technology has become successful and popular at some mines of The Singareni Collieries Company Limited, Andhra Pradesh and Chirimiri colliery of South Eastern Coalfields Ltd, India. Thick seam ranging from 7 to 15 m is developed and a panel is found suitable to extract within incubation period. Diagonal line of extraction is followed in sequence to extract total thickness of coal by ring holes drilling and blasting and by using of remote control load haul dumpers (LHD). This method of pillar extraction in a rib-less method does not require any goaf edge support.

1.1 Problems of Strata Behaviour Characteristics with Respect to Indian Coal Seams

The BG working is not suitable for higher degree of gassiness with irregular seam characteristics. BG method if applied in irregular seam will cause unblasted waste rock mixture with the coal. Overriding of galleries may be a regular phenomenon if a attention in not paid on the extraction pattern in time. Chances of air blast with consequent possibilities of spontaneous heating in the goaf seem to be a major problem in this working. The increasing roof pressure creates major difficulties with setting of goaf edge breaker line support. This leads to more chances of coal losses with a significant reduction in overall performance of the mine.

1.2 Objectives of the Project

The Blasting Gallery operation in a mechanized underground mine system depends upon many decisions influenced by the geo-technical parameters which are often interspersed with inherent strata configurations. The present study has been aimed to examine thoroughly BG method operational systems in Indian geo-mining conditions such as:

- Study of roof convergence with respect to face advancement during different stages of extraction of coal in Blasting Gallery panels in SCCL mines.
- Simulation of field conditions in the numerical model generated using FLAC.
- Interpretation of strata behaviour through numerical modeling using FLAC.

CHAPTER 2

LITERATURE REVIEW

2. LITERATURE REVIEW

Blasting Gallery technology, the successful and popular method of extraction with a given set of input, has been a good source of underground production in India. The moderate to high overburden always poses a problem to tackle with the strata in day to day’s work. The extensive literature survey has been a prime part of this system and has given a priority out of all subroutines considered here for this purpose. Blasting gallery method is a unique technique successfully developed in France, where it has been practiced in virgin thick seams in Carmaux colliery. Blasting gallery method was earlier experienced at East Katras colliery of the Bharat Coking Coal Limited and Chora colliery of the Eastern Coalfields Limited of the coal India limited. Overriding at East Katras colliery and loss of supports as well as coal had put some question marks on its further application in Indian coal Industry. But this technology has become successful and popular at some mines of The Singareni Collieries Company Limited, Andhra Pradesh and Chirimiri colliery of South Eastern Coalfields Ltd, India. (Singh R.D. 1998)

2.1 Blasting Gallery Method

In this method a seam is developed into panels of about 100 m x 50 m. From the main headings rooms are driven to the full width of the panel and the coal between the rooms is blasted down to the full thickness of the seam and loaded by remotely controlled loaders. The layout of a panel for working by Blasting Gallery method is shown in Figure 2.1. The life of the rooms should be kept as short as possible so that they do not undergo excessive convergence and the movement of the vehicles is not rendered any difficulty.

The advantage of this system of mining is that, it makes it possible to win narrow panels or larger panels in which the seam conditions are unsuitable tor a longwall face. It does not require highly experienced workers as a longwall face with 'Soutirage' working .It requires substantially less investments than those required for a longwall with soutirage working and the equipment required i.e., heading machines or jumbos and LHD can be easily transferred to other roadways if the method is unsuccessful. Thick seams up to 15 m in thickness can be extracted in one pass with percentage extraction ranging from 65 to 85%.The method is highly flexible in that in a district with several units in operation, even if one of the units is under breakdown, production from the district will continue to come. The time required for preparation of a panel in relation to the total life of the panel if less than with other mechanized methods. (Majumdar S et al. 2011)

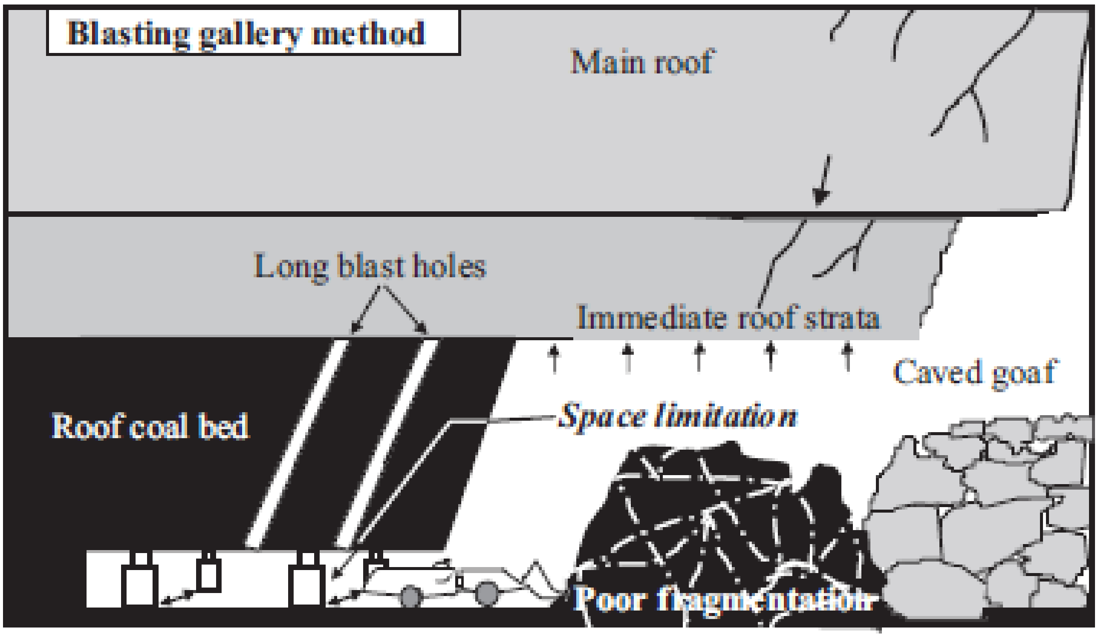


Figure 2.1 Blasting Gallery Method (Jayanthu S. 2005)

2.2 Strata Behaviour

Strata control or roof control implies the control of strata to facilitate mining operations to be done efficiently and safely. The strata in the face and in the adjoining area require attention so that no uncontrolled failure of the ground occurs.

2.3 Theories of Mechanics of Strata Behaviour

The various common theories in this regard include the following:

- Dome or Arch Theory
- Beam or Plate Theory
- Soil Mechanics Theory
- Pseudo Plastic Theory
- Hypothesis Based on Law of Deformation
- Dynamic Rock Pressure Theory

2.4 Strata Pressure Redistribution in Blasting Gallery Workings

The development of pressure on pillars appears to be dependent on three factors, viz. Depth from surface, Area of development, Ratio of areas of bord centers to area of pillar formed. Full pressure on pillars due to the weight of overlying strata is experienced much sooner in shallow mines than in deep mines. Research from SA report that the maximum pressure is exerted about the center of the area developed, but is less on completed pillars, from this point towards the direction of the solid undeveloped coal. The degree of pressure at the center of any solid any developed area, nearly circular in plan, is approximately represented by thick strata above the seam equal to radius multiplied by ratio of area of pillars to that of the bord centers. Where the completed pillars in development are circular, the pressure operating within that area may be likened in a diagram to a cone or parabola. If the developing are is square in plan, the pressure is likened to be a pyramid.

2.5 Convergence in Blasting Gallery Workings

Convergence in development headings is influenced by the nature of roof and floor and width of headings. Weak roof and floors and wide galleries gives more convergence than that with strong roof and floor and narrow galleries. In Indian coal mines, as the immediate roofs are generally strong, hardly any convergence is observed. In depillaring and sequence of extraction, besides the nature of roof and floor and whether the goaf is caved or stowed. Because of the stooks and remnants left, the convergence does not follow any predictable pattern. When the goaf is stowed, the convergence is less than what would occur if the goaf is caved.

2.6 Indian Scenario

Banerjee (2006) in his paper stated that Blasting Gallery method has been a popular method in SCCL and 800-to 1000 tpd production has been obtained from many panels at a much lower investment than in PSLW faces.

Jayanthu (2005) has given his view that in caving panels, the nether roof strata in goaf before first major fall behaves as a simply supported beam fixed one end to the panel barrier with the goaf edge support/rib adjoining the working face acting as other end. In view of extensive qualitative observations and theory, the roof strata fail due to tensile fractures, while the sides suffer with shear failures. Hence, using been analogy, critical span for a competent layer clamped all around under uniformly distributed load can be derived on the basis of tensile strength of the competent layer. However, the end conditions of clamped beam with change to an edge supported beam due to development of tensely cracks on the upper surface near the ends of the beam.

Ray, Singh and Banerjee (2005) assessed that the direction and magnitude of horizontal stress has a significant influence upon caving of the rock mass. Horizontal stress impedes the failure of rock mass as it provides a confinement to the rock mass subjected to underground loading condition. Vertical stresses add to the development of tensile and sheer stresses. In deeper workings where the vertical stress in considerably high, the roof cavability is better compared to a shallow working having similar roof lithology.

Satyanarayana et al (2005) expressed their views that with decrease in the distance of the monitoring point from the goaf edge, convergence is increased. This convergence attains its maximum value at the goaf edge. Similar phenomena also happen for strata load.

Venkatanarayana (2004) stated that normally the goaf of long wall or of BG panel is completely packed in the middle of the goaf and along the barrier goaf consolidation will be less. Panel size had been reduced from 25,000 m2 to 16,000 m2. To induce the main fall several induced basting were under taken in the panel. He found occurrences of several major falls before closing of the panel.

While monitoring of strata movement during underground mining of coal, Rajendra Singh et, al (2004) found that the value of mining induced stress over pillar and roof to floor convergence during depillaring, generally, increases with decrease in distance of the observation station with respect to the line of extraction. Similarly the values of other parameters like bed separation, load on support etc. were also influenced by the face advance of a depillaring panel.

Rajendra Singh et, al (2004) stated the most challenging job during implementation of the blasting gallery method was to provide effective support to the high roof after wining of the roof coal and they solved this problem simply by introducing cable bolting as a support system for the high roof as well as for the overlying coal bed.

Jayanthu (2001) observed while investigating strata behavior in a Blasting Gallery panel during extraction of bottom section pillars at greater depth that the rate of convergence reached a maximum per value of about 3.5mm/ day during major roof fall in the panel. Increasing rate of convergence may be attributed to the roof falls in the goaf associated with about 60% to 80% filling of the goaf.

NIRM (2000) in their report of strata monitoring in BG panel at GDK-10 Incline described that the area of extraction at the time of major roof falls was more than 12,000m2, without any damage to the advance workings. Due to the influence of the barrier up to 25m alongside, in general, induce blasting near the barrier may not contribute to the major roof fall. While optimizing blast design and charge loading parameters in coal for ring hole blasting and in stone for Induce blasting in degree –1 seam for Blasting Gallery method.

While Studying of weathering action on coal pillars and its effects on long term stability, Biswas and Peng (1999) observed that if a coal pillar is exposed to moisture and if it has a parting layer in it, structural deterioration takes place over time. This deterioration reduces the load carrying capacity of the pillar.

During investigation in to the strata behavior of panel H in East Katras colliery, Raju et al (1998) indicated the failure of the parting above junction of bottom section due to high tensile abutment stresses and also suggested that the galleries of the two sections must be superimposed and high support resistance is needed in junctions in top and bottom sections.

Samantha (1997) described in his paper that at Chora 10-pit colliery during working by Blasting Gallery method the immediate roof was very brittle and quite difficult to control above a freshly blasted area and this problem has been solved by leaving 0.6 m coal at roof.

Venkateswarlu and Raju (1993) stated that roof stability is a function of several factors such as the inherent physico-mechanical character of the rock, presence of geological anomalies, method of working and the mining environment and design of roof supports in coalmines based on geomechanical classificatory.

Raju (1986) observed that first main fall took place after an area of exposure of 6600 sq.m in goaf. Subsequently the main falls took palace regularly after every 5310 sq.m to 11000 sq.m of area of exposes. At no time during the 18 months extraction period the support system (roof bolts, channels and props), used in conjunction with LHDs had given any untoward experience.

2.7 International Scenario

Khair and Peng (1985) expressed their views that it is very unlikely to have pillar failure due to vertical stress alone. The major factors probably were the rupture of the roof due to abrupt change in the topography of the coal seam and possible high horizontal stress.

From the studies conducted by Wen-Xiu L, Lan-Fang D, Xiao-Bing, H and Wen L (2007) the prediction of ground surface movements was found to be important problem in rock and soil mechanics in the excavation activities. Based on results of the statistical analysis of a large amount of measured data in underground excavation engineering, the fuzzy genetic programming method (FGPM) of ground surface movements is given by using the theory of fuzzy probability measures and genetic programming (GP).

Unver and Yasitli (2006) stated that top coal, caving behind the face is the key factor affecting the efficiency of production at thick coal seams. Their results included that in order to decrease dilution and increase extraction ratio and production efficiency, the top coal should be as uniformly fractured as much as possible. Hence, an efficient and continuous coal flowing behind the face can be maintained. A special pre-fracture blasting strategy just sufficient enough to form cracks in the top coal is suggested by means of comparing results from numerical modelling.

Jialin Xu and Minggao (2005) observed that rock strata move upward from the coal seam to the surface by groups and the breakage and movement of key stratum determine the dynamic process of rock strata movement.

Cox (2003) suggested that the ground forces generated by a properly installed and tensioned mine roof truss assembly can provide permanent mine roof support, even in severe ground conditions. This can be accomplished either by direct suspension of the rock loads within the potential failure zone above the mine opening or by indirect reinforcement of the natural rock arch that tends to form within the immediate mine roof.

Tekook and Keune (1999) declared that in Indian deposits with shallow depths and thick sandstone in the roof, strata control has the same importance as in deep mining. They have stressed on measurements and observations in galleries and faces, inference of behaviour of support and strata, verification of the planning and developing the prediction methods.

Garratt (1999) stated that the stresses acting on underground workings are pre-mining stresses, interaction induced stresses caused by nearby workings and stresses caused by current excavation.

CHAPTER 3

NUMERICAL MODELLING

3. NUMERICAL MODELLING

3.1 Overview

FLAC[16] is a two-dimensional explicit finite difference program for engineering mechanics computation. This program simulates the behavior of structures built of soil, rock or other materials that may undergo plastic flow when their yield limits are reached. Materials are represented by elements, or zones, which form a grid that is adjusted by the user to fit the shape of the object to be modeled. Each element behaves according to a prescribed linear or nonlinear stress/strain law in response to the applied forces or boundary restraints. The material can yield and flow and the grid can deform and move with the material that is represented. The explicit, Lagrangian calculation scheme and the mixed-discretization zoning technique used in FLAC ensure that plastic collapse and flow are modeled very accurately. Because no matrices are formed, large two-dimensional calculations can be made without excessive memory requirements. The drawbacks of the explicit formulation are overcome to some extent by automatic inertia scaling and automatic damping that do not influence the mode of failure.

Though FLAC was originally developed for geotechnical and mining engineers, the program offers a wide range of capabilities to solve complex problems in mechanics. Several built-in constitutive models that permit the simulation of highly nonlinear, irreversible response representative of geologic, or similar, materials are available. In addition, FLAC contains many special features including:

- Interface elements to simulate distinct planes along which slip and/or separation can occur
- Plane-strain, plane-stress and axisymmetric geometry modes
- Groundwater and consolidation models with automatic phreatic surface calculation
- Structural element models to simulate structural support
- Extensive facility for generating plots of virtually any problem variable
- Optional dynamic analysis capability
- Optional viscoelastic and viscoplastic models
- Optional thermal and thermal coupling to mechanical stress and pore pressure modeling capability
- Optional two-phase flow model to simulate the flow of two immiscible fluids through a porous medium

3.2 Problem Solving With FLAC

The problem is solved by using FLAC in the following sequence of steps :

- Grid generation
- Boundary and initial conditions
- Loading and sequential modeling
- Choice of constitutive model and material properties
- Ways to improve modeling efficiency
- Interpretation of results

3.3 Recommended Steps For Numerical Analysis In Geomechanics

The recommended steps for solving a real life situation can be modelled as follows:

Step 1: Define the objectives for the model analysis

Step 2: Create a conceptual picture of the physical system

Step 3: Construct and run simple idealized models

Step 4: Assemble problem-specific data

Step 5: Prepare a series of detailed model runs

Step 6: Perform the model calculations

Step 7: Present results for interpretation

3.4 Steps for Numerical Modelling

Step 1: Define the Objectives for the Model Analysis

The level of detail to be included in a model often depends on the purpose of the analysis. For example, if the objective is to decide between two conflicting mechanisms that are proposed to explain the behavior of a system, then a crude model may be constructed, provided that it allows the mechanisms to occur. It is tempting to include complexity in a model just because it exists in reality. However, complicating features should be omitted if they are likely to have little influence on the response of the model, or if they are irrelevant to the model's purpose.

Step 2: Create a Conceptual Picture of the Physical System

It is important to have a conceptual picture of the problem to provide an initial estimate of the expected behavior under the imposed conditions. Several questions should be asked when preparing this picture. Considerations will dictate the gross characteristics of the numerical model, such as the design of the model geometry, the types of material models, the boundary conditions, and the initial equilibrium state for the analysis. They will determine whether a three-dimensional model is required, or if a two-dimensional model can be used to take advantage of geometric conditions in the physical system.

Step 3: Construct and Run Simple Idealized Models

When idealizing a physical system for numerical analysis, it is more efficient to construct and run simple test models first, before building the detailed model. Simple models should be created at the earliest possible stage in a project to generate both data and understanding. The results can provide further insight into the conceptual picture of the system. Step 2 may need to be repeated after simple models are run. Simple models can reveal shortcomings that can be remedied before any significant effort is invested in the analysis.

Step 4: Assemble Problem-Specific Data

The types of data required for a model analysis include:

• Details of the geometry (e.g., profile of underground openings, surface topography, dam profile, rock/soil structure)

• Locations of geologic structure (e.g., faults, bedding planes, joint sets)

• Material behavior (e.g., elastic/plastic properties, post-failure behavior)

• Initial conditions (e.g., in-situ state of stress, pore pressures, saturation)

• External loading (e.g., explosive loading, pressurized cavern)

Since, typically, there are large uncertainties associated with specific conditions in particular: state of stress, deformability and strength properties, a reasonable range of parameters must be selected for the investigation. The results from the simple model runs can often prove helpful in determining this range, and in providing insight for the design of laboratory and field experiments to collect the needed data.

Step 5: Prepare a Series of Detailed Model Runs

Most often, the numerical analysis involves a series of computer simulations that includes different mechanisms under investigation and span the range of parameters derived from the assembled database. When preparing a set of model runs for calculation, several aspects, such as those listed below, should be considered:

Step 6: Perform the Model Calculations

It is best to first make one or two model runs split into separate sections before launching a series of complete runs. The runs should be checked at each stage to ensure that the response is as expected. Once there is assurance that the model is performing correctly, several data files can be linked together to run a complete calculation sequence. At any time during a sequence of runs, it should be possible to interrupt the calculation, view the results, and then continue or modify the model as appropriate.

Step 7: Present Results for Interpretation

The final stage of problem solving is the presentation of the results for a clear interpretation of the analysis. This is best accomplished by displaying the results graphically, either directly on the computer screen, or as output to a hardcopy plotting device. The graphical output should be presented in a format that can be directly compared to field measurements and observations. Plots clearly identify regions of interest from the analysis, such as locations of calculated stress concentrations, or areas of stable movement versus unstable movement in the model. The numeric values of any variable in the model should also be readily available for more detailed interpretation by the modeler.

The software uses a particular module for carrying out the operations sequentially. Before producing the final outputs, it undergoes several steps and reconsiders the parameters. If the results are found satisfactory, then the final result is displayed and if not, then re modeling by changing parameters is done. The entire procedure is shown in the form of a flowsheet in Fig.3.1.

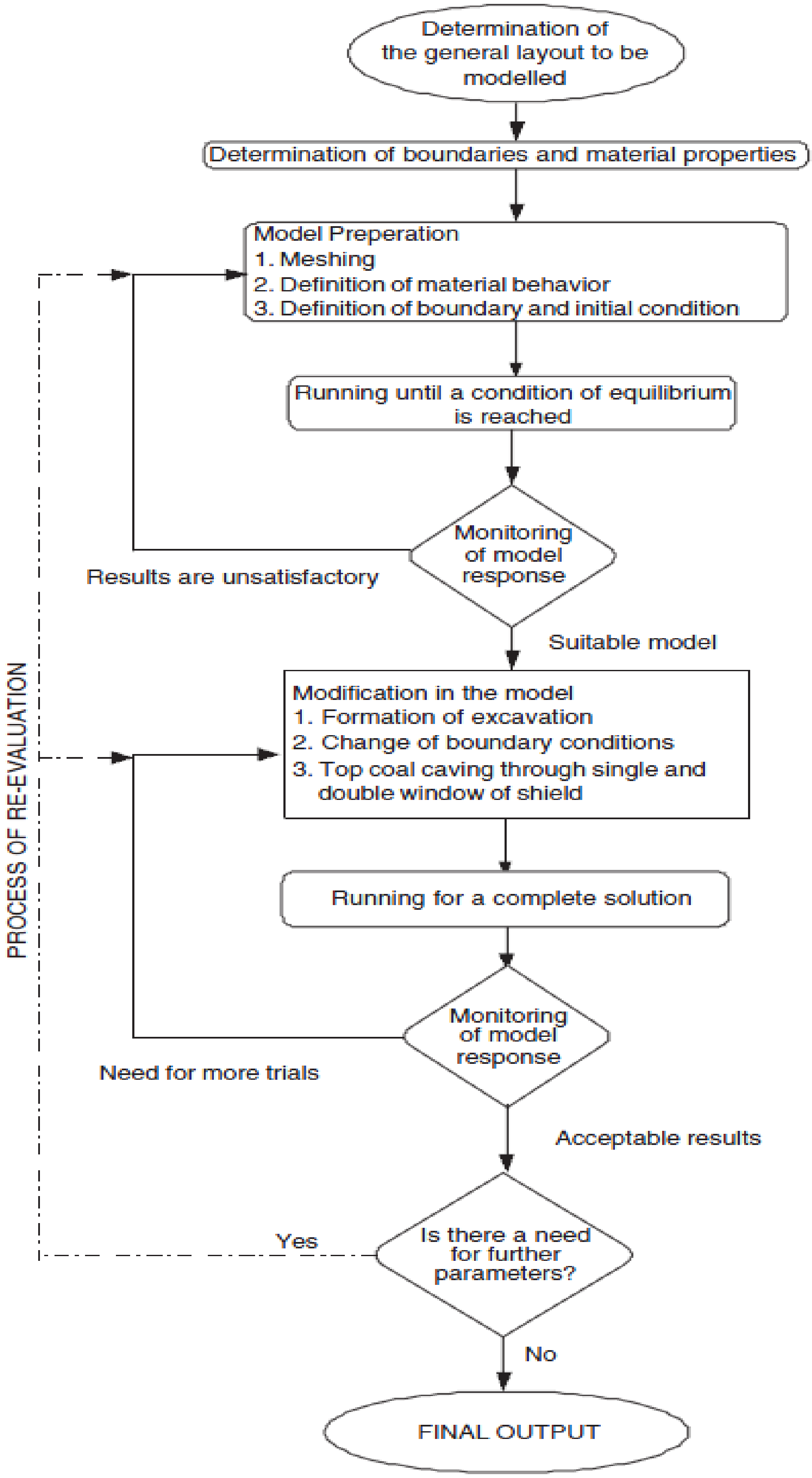


Fig.3.1 A general flow sheet of modelling procedure

CHAPTER 4

METHODOLOGY

4. METHODOLOGY

The extensive literature survey has shown that the BG method has been a proven technology with a high potential to augment the production and productivity. The cross section of the survey has also pointed out few areas which need to be stressed upon for the betterment of the system. The pin pointed areas includes impact of depth on first major fall, convergence trend, extent of convergence in a room, overburden stress pattern or behaviour etc. In order to fulfill the study as well as findings of the research work as cropped out of the survey, the following steps were taken:

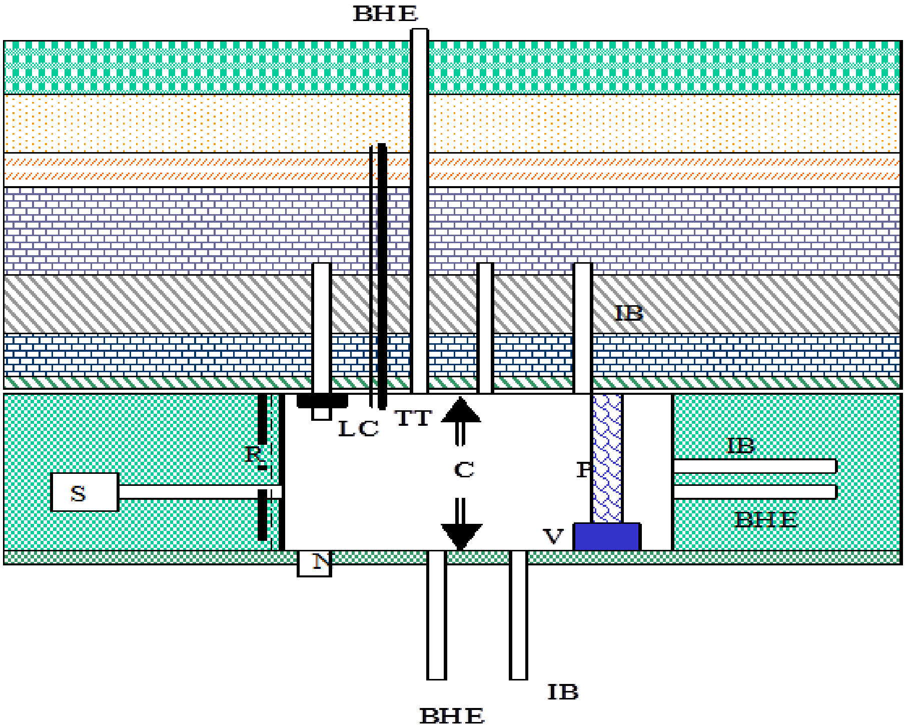
- An underground mine practicing BG method of working was chosen for detailed study in this context.
- Strata was monitored with the help of electronic gadgets or instruments.
- Convergence data, Day wise reading and the position of convergence station with respect to Goaf Edge Distance (GED) were recorded for every room of the panel.
- The Geotechnical conditions of the mine were simulated in models.
- Numerical Modeling was done and the strata behaviour was predicted
- Results were validated by comparing the actual behaviour of strata with the results of Numerical Modelling at various stages of extraction.

4.1 Selection of Sites

Study was conducted in the mines practicing Blasting Gallery method of working. As such mine of Singareni Collieries Company Limited has been selected. For the study, GDK No. 10 incline’s 3A panel was selected. The brief description of the panel is shown separately in Appendix.

4.2 Instrumentation

The instruments like telescopic convergence indicators were fixed in predetermined places to get convergence. All the instruments as used for this purpose were calibrated prior to use in the field. The fig.4.1 shows the typical instrumentation setup for general study of strata behaviour in coal mines. The usage or installation points of the instruments were chosen judiciously.



Legends
IB – Instrumented Bolt
BHE–Bore Hole Extensometer
M – Magnetic ring Anchor
C – Convergence Indicator
S – Stress Capsule
LC – Anchor Load Cell
P – Prop Support
R–Remote Convergence Indicator in a grove
TT – Tell Tale Instrument

Fig.4.1 Typical Instrumentation Set Up for Strata Behaviour Study

Fig.4.2 Convergence Indicator

The figure 4.2 shows the instrument used to measure convergence in the field. The pointer pointing to the number was initially noted while installing in the field and the reading was taken once every day. The difference between the initial and final readings gives the convergence observations for that day. The figure 4.3 shows the insitu measurement of convergence by using the convergence indicator instruments.

Fig.4.3In Situ Measurement Of Convergence

Convergence indicators were installed in 66 AL, 66 BL, 67L, 67AL, 67 BL, 68L, 68AL, 68BL, and 70 Level in the 41 to 43 dips at an interval of 10 m along the levels. The figure 4.4 shows the instrumentation layout of the GDK 10 Incline's BG-3A panel practicing the Blasting Gallery Method.

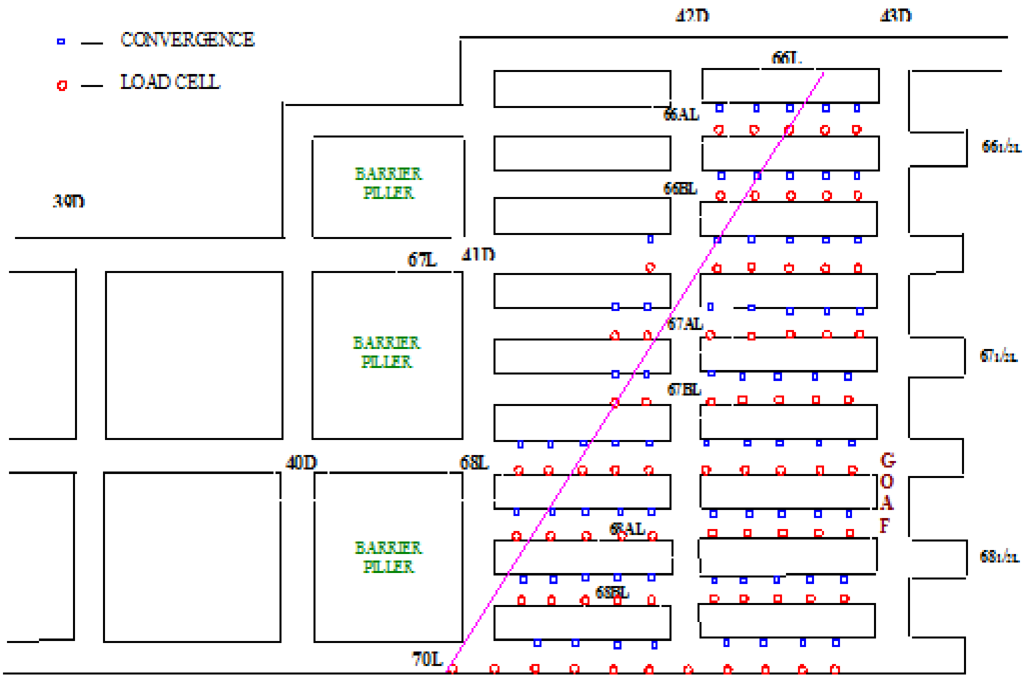


Fig.4.4 Instrumentation Layout in BG 3A Panel of GDK 10 Incline, SCCL

4.3 Modelling parameters

Depillaring process in this numerical method includes different stages of division of pillars in to stooks and extraction of stooks up to full seam thickness leaving some ribs in the goaf. For two dimensional representation of full seam extraction in a seam, vertical section with four galleries in an idealised panel was selected. A few parameters were kept constant for the model, e.g. width of the pillar, development gallery, split gallery as 50 m, 4.2 m and 4.2 m respectively. In the first stage of extraction, splits of 4.2 m width were provided. And the second, third and fourth stages of extraction include high opening up to full seam thickness with formation of ribs in the goaf. Strata behaviour via convergence or deformation in the roof of galleries in these conditions was studied through numerical models.

Numerical models with different configuration of openings representing the range of parameters in the field experiment trials were used. The parameters considered from the mine conditions for convergence were considered to be of 11m seam thickness at a depth cover of 350m.

4.4 Sequence of Modelling

1. Development of the seam with three pillars and four galleries of 50m and 4.2m width respectively.
2. Development of two splits of 4.2m width in one Pillar.
3. Development of two splits of 4.2m width in second Pillar and extraction of Stook 1 to full thickness.
4. Development of two splits of 4.2m width in third pillar and extraction of Stook 2 to full thickness.
5. Extraction of Stook 3 to its full thickness.
6. Extraction of Stook 4 to its full thickness.
7. Extraction of Stook 5 to its full thickness.

To reduce the time to solve the model, the dimensions of the mesh elements increase geometrically from the model to its outer edges. The model has plate elements with nodes and the problem domain consist of approximate boundary conditions and grid pattern for 350 meter depth cover with development into extraction in plain strain conditions with Mohr Coulomb material.

The top of model is free to move in any direction, and the bottom edge of the model is restricted from moving vertically. Roller type boundary conditions for all the models are placed along two edges of the models. In the absence of the in-situ stress measurement in the coal field, the following norms were adopted for estimation of in-situ stress field prior to the excavation of the area.

$$\text{Vertical stress} = \rho \times H$$

$$\text{Horizontal stress} = 3.75 + 0.015\,H$$

Where,

ρ = specific weight of the overlying rock mass and

H = depth cover

The model has induced internal stress that simulates gravity loading. To generate pre-mining conditions before adding the mine openings to the input, the model goes through an initial analysis to generate the insitu stresses. Gravitational and horizontal loading are forced on the other two surfaces in order to account for insitu stresses. The displacements are reset to zero and the mine openings were added. The model was then reanalyzed to obtain the final deformation. The properties of Coal and Sandstone are shown in the Table no. 4.1.

4.5 Experimentation

4.5.1 Sample Preparation

For the purpose of determining the rock mass parameters, it is highly essential to go for experimentation in laboratory. For this purpose, Coal samples were brought from the Grab samples from the BG Panel of Singareni Collieries Company Limited, Ramagundam.

After obtaining the requisite amount of coal sample, the core is created using the coring machine in the Geomechanics Laboratory of Mining Department of National Institute of Technology, Rourkela. Three core samples were prepared, of 54mm diameter and 108mm in length, having an L/D ratio of 2:1. This core sample is tested for obtaining the essential parameters.

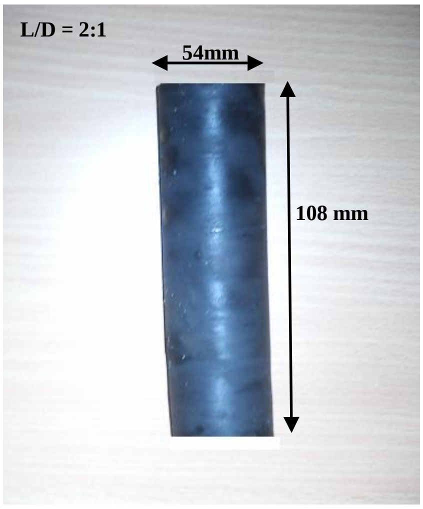


Fig. 4.5 Coal Core Sample for Triaxial Testing

4.5.2 Triaxial Testing[17]

A triaxial shear test is a common method to measure the mechanical properties of many deformable solids, especially soil and rock, and other granular materials or powders. Although the name triaxial test suggests that the stresses would be different in three directions, this is not true in the test as is usually done. In this test with oil or water as confining medium, the confining pressures are equal in all directions. Only in a true triaxial test the stresses in all three directions can be different (i.e. $\sigma1 \neq \sigma2 \neq \sigma3$).

The principle behind a triaxial shear test is that the stress applied in the vertical direction (along the axis of the cylindrical sample) can be different from the stresses applied in the horizontal directions perpendicular to the sides of the cylinder, i.e. the confining pressure). In a homogeneous and isotropic material this produces a non-hydrostatic stress state, with shear stress that may lead to failure of the sample in shear. In homogeneous and anisotropic samples failure may occur due to bending moments and, hence, failure may be tensile. Also combinations of bending and shear failure may happen in inhomogeneous and anisotropic material.

From the triaxial test data, it is possible to extract fundamental material parameters about the sample, including its angle of shearing resistance, apparent cohesion, and dilatancy angle. These parameters are then used in computer models to predict how the material will behave in a larger-scale engineering application. Different types of triaxial tests include Consolidated Drained, Consolidated Undrained and Unconsolidated Undrained.

The following is a basic outline of the triaxial test procedure:

- The specimen is a cylindrical sample normally 54 mm in diameter by 108 mm length. The sample is generally compacted in the laboratory (Fig. 4.5).
- The specimen is enclosed vertically by a thin "rubber" membrane and on both ends by rigid surfaces (platens) as sketched in Figure 4.7.
- The sample is placed in a pressure chamber and a confining pressure is applied (σ_3) as sketched in Figure 4.8.
- The deviator stress is the axial stress applied by the testing apparatus (σ_1) minus the confining stress (s3). In other words, the deviator stress is the repeated stress applied to the sample. These stresses are further illustrated in Figure 4.9.
- The resulting strains are calculated over a gauge length, which is designated by "L" (Figure 4.10).
- Basically, the initial condition of the sample is unloaded. When the deviator stress is applied, the sample deforms, changing in length as shown in Figure 4.11. This change in sample length is directly proportional to the stiffness.

Fig. 4.6 Laboratory Setup for Triaxial Testing

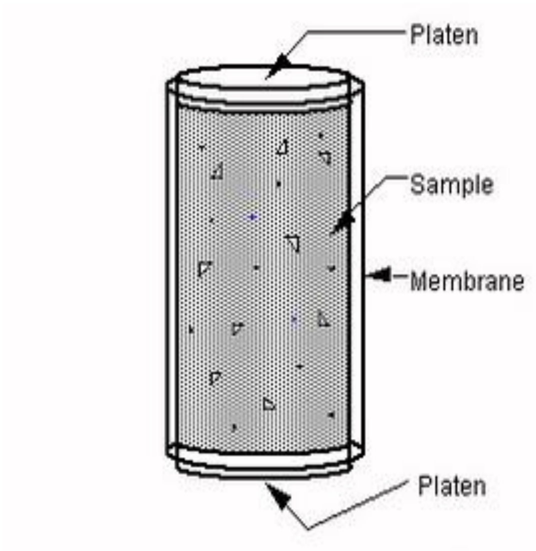


Fig. 4.7 Enclosure of Triaxial Specimen

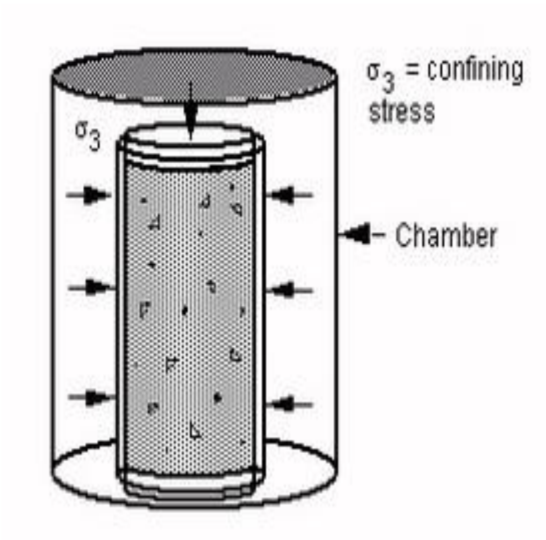


Fig. 4.8 Triaxial Specimen in Pressure Chamber

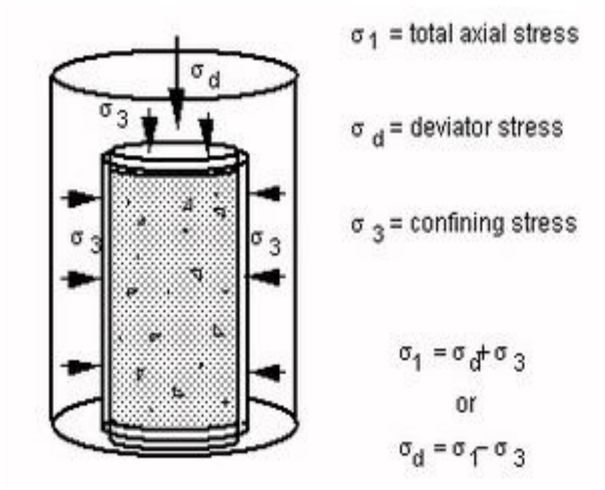


Fig. 4.9 Stresses Acting on Triaxial Specimen

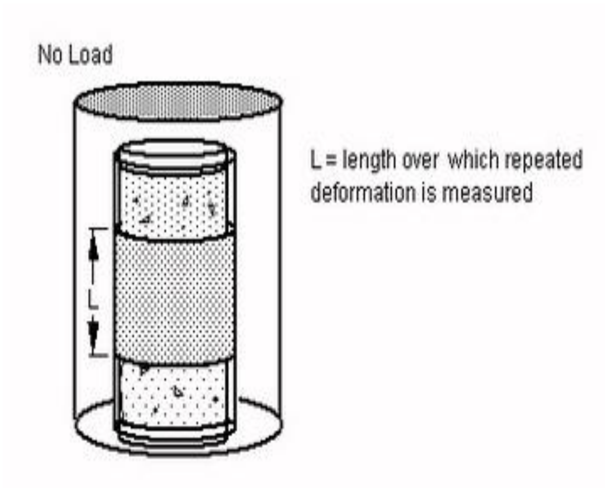


Fig. 4.10 Gage Length for Measurement of Strain on Triaxial Specimen

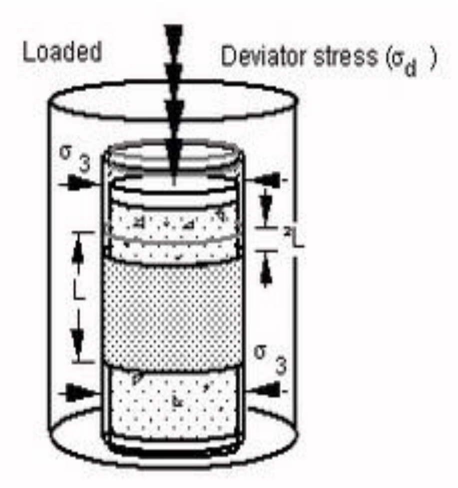


Fig. 4.11 Deformation of Triaxial Specimen Under Load

The results obtained from the triaxial testing include the following:

Table 4.1: Results of Triaxial Test

Core Sample No.	Confining Stress (σ_3 in MPa)	Deviator Stress (σ_1 in MPa)
1	0	22.0
2	2	32.0
3	4	41.5

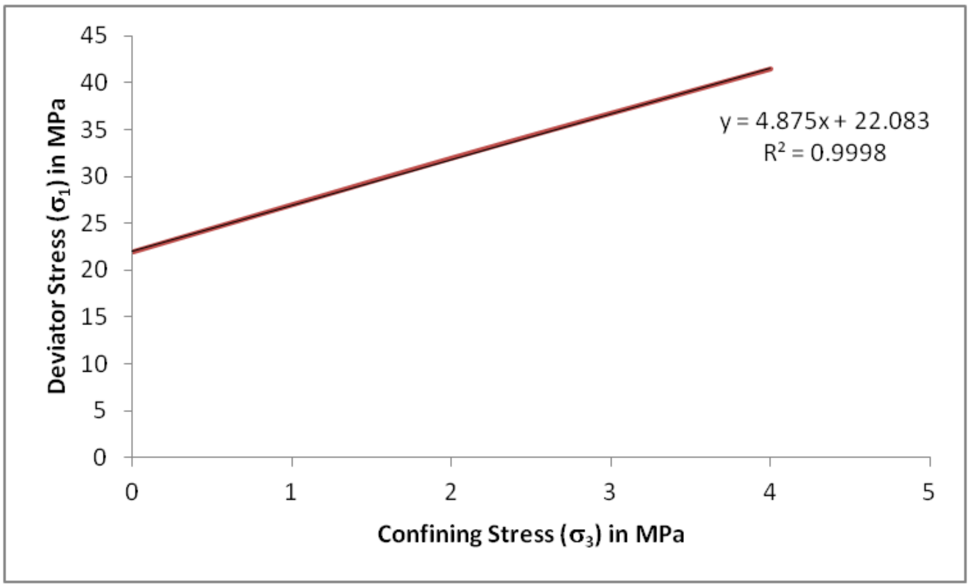


Fig. 4.12: Deviator vs. Confining Stresses

The results obtained from the laboratory testing that of σ_1 & σ_3 was used as an input parameters in the RocLab software and the following results were obtained.

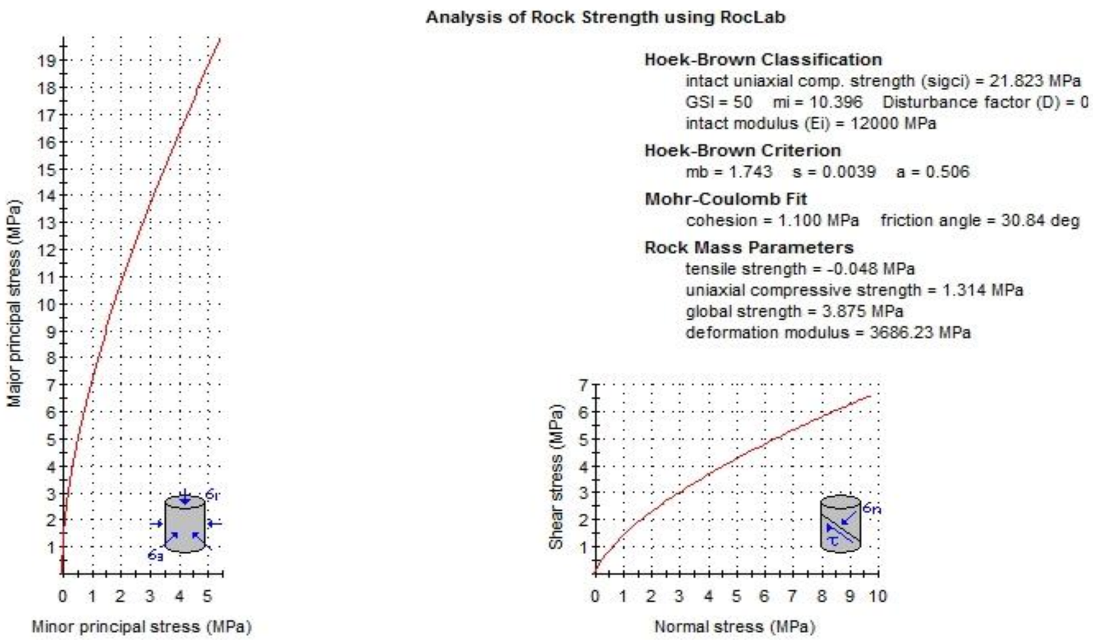


Fig. 4.13 Triaxial Results using RocLab software

One of the major obstacles which is encountered in the field of numerical modeling for rock mechanics, is the problem of data input for rock mass properties. The usefulness of elaborate constitutive models, and powerful numerical analysis programs, is greatly limited, if the analyst does not have reliable input data for rock mass properties. The obtained sigma values from the triaxial test is taken by the RocLab[18] - rock mass strength analysis software as the Lab Analysis data and follows a particular failure criterion and provides the results of the rock mass parameters, which are used in the numerical models for prediction of the strata behaviour.

Table 4.2: Properties of Coal

Property	Coal
Density (D)	1427 kg/m^3
Tensile Strength (T)	0.32 MPa
Cohesion (c)	1.1 MPa
Friction Angle (ɸ)	30.84^0
UCS	1.314 MPa

CHAPTER 5

FIELD INVESTIGATION

5. FIELD INVESTIGATION

To form the data base, the information of Blasting Gallery method of work have been collected and processed through some stages. The respective mine was visited and experience is gained on the system of operation. Data has been collected from instruments installed in the BG panels and through log books and registers of the mine concerned. The data has been checked and authenticated by the strata control officers of those mines.

Data was collected from office records maintained daily shift wise basis. Different strata monitoring instruments and their functions were taken from manuals supplied by manufacturers. The data of natural falls, induced blasting etc. were also collected from mine records and they were checked with respective mine strata monitoring in charge. Again the data collected were verified at Regional Strata Control Cell. The strata monitoring data and different information collected from Blasting Gallery panel was synthesized to evaluate the behaviour of strata.

5.1 Strata Behaviour Observations in BG Panel 3A, GDK 10 Incline, SCCL

For safe operation of depillaring by Blasting Gallery method, it was mandatory to record convergence at each room daily. It was major indicative of the movement of strata and gives well in advance indication of impending fall in goaf or weighting on pillars. Telescopic convergence indicator was used to observe the convergence behaviour. Data collected were shown in the following tables for BG 3A panel, GDK 10 incline of Singareni Collieries Company Limited.

Convergence stations were installed at about 10 m interval along the levels and sublevels 70L, 68BL, 68AL, 68L, 67BL, 67AL, 67L, 66BL and 66AL in the BG panel 3A. Convergence observations at C9-70L indicated no perceptible roof movement. About 4 mm cumulative convergence was noticed at this station. Less convergence at this station maybe attributed to the barrier effect.

Comparatively more convergence of workings was observed at C9-68BL. About 18 mm cumulative convergence was noticed at this station up to the end of 31st August'2011. Sounds were observed in 68BL followed by stone fall in pre-shift on 10th August 2011, with convergence of 1 mm only at the station C9 – 68BL.At 68AL, it was observed that station C-7 has the cumulative convergence of 22 mm. Maximum rate of convergence of about 2 mm/day was observed when the station was nearer to the goaf edge i.e., 4 m (Table-5.3). Sounds observed and followed by stone fall in pre-shift on 10th August 2011.

At station C6 in 68L about 38 mm cumulative convergence was noticed up to the end of 31st August'2011. Maximum convergence about 3 mm was observed when the station was nearer to the goaf edge i.e., 3 m. Stone fall took place on 20th August 2011 in third shift. The station C5-67AL was installed on 05th August'2011 at a distance of about 16 m from the goaf edge and maintained up to 31st August 2011. Maximum of 13 mm cumulative convergence was observed for the last five days as it was very nearer to the goaf edge. Total cumulative convergence at this station was 23 mm (Table-5.3). A fall took place after indicating 3 mm convergence on 27th August 2011 in first shift.

The station C3-67AL was installed on 10th August'2011 at a distance of about 22 m from the goaf edge and it is observed that maximum daily convergence recorded when the goaf edge was 8 m from the station. Total cumulative convergence at this station is 17 mm. Maximum convergence was observed when station nearer to goaf edge. Total cumulative convergence at the station C4-66AL is 26 mm. Maximum convergence observed when station nearer to goaf edge.

The station C6-67AL was installed on 1st October'2011 at a distance of about 4 m from the goaf edge and it was observed that maximum daily convergence recorded when the goaf edge was 16 m from the station. Total cumulative convergence at this station was 32 mm (Table-5.5). Maximum convergence was observed when station nearer to goaf edge. At the station C9-68AL, it was observed that maximum daily convergence recorded when the goaf edge was 10 m away from the station. Total cumulative convergence at this station was 27 mm. Maximum convergence was observed when station nearer to goaf edge. Total cumulative convergence at C11-67L was 36 mm. Total cumulative convergence at C10-66AL was 15 mm.

Table 5.1: Convergence Observation up to the end of the Month of June –11

	Location	Convergence observation				
		Cumulative convergence		Max. convergence change in a day		Stone fall details
		(0.5m)	(2.5m)	(0.5m)	(2.5m)	
1.	70L	17mm (C4)	11mm (C4)	2mm (C4) on 06-06-2011 GED 2 m	2mm (C4) on 06-06-2011 GED 2 m	Natural fall occurred on 07-06-11 and induced blasting done on 17-06-11.
2.	68BL	***26mm (C4)***	***16 mm (C4)***	2.0 mm(C4) on 28-06-11 GED 2 m	2.0 mm(C4) on 28-06-11 GED 2 m	Natural fall occurred on 07-06-11 and on 30-06-11.
3.	68AL	28 mm (C3)	20 mm (C3)	2.0 mm(C3) on 29-06-11 GED 4 m	2.0 mm(C2) on 29-06-11 GED 4 m	Natural fall occurred on 30-06-11.
4.	68L	26mm (C3)	18mm (C3)	2.0 mm(C3) on 25-06-11 GED 4 m	2.0 mm(C3) on 25-06-11 GED 4m	Induced blasting done on 26-06-11 and Natural fall occurred on 27-06-11.
5.	67BL	23 mm (C2)	18 mm (C2)	2.0 mm(C2) on 26-06-11 GED 2 m	2.0 mm(C2) on 26-06-11 GED 2 m	Natural fall occurred on 27-06-11 and on 29-06-11.
6.	67AL	12 mm (C1)	9 mm (C1)	2.0 mm(C1) on 17-06-11 GED 5m	1.0 mm(C1) on 26-06-11 GED 2 m	No fall

7.	67L	11 mm (C1)	8mm (C1)	1.0 mm(C1) on 26-06-11 GED 4 m	1.0 mm(C1) on 30-06-11 GED 2 m	No fall

GED- Goaf Edge Distance

Table 5.2: Convergence Observation Up To the End of the Month of July –11

	Location	**Convergence observation**				
		Cumulative convergence		**Max. convergence change in a day**		**Stone fall details**
		(0.5m)	**(2.5m)**	**(0.5m)**	**(2.5m)**	
1.	70L	9mm (C6)	8mm (C6)	1mm(C6) on 19-07-2011 GED 6m	1mm(C6) on 19-07-2011 GED 2 m	Natural fall occurred on 18-07-11 and on 19-07-11.
2.	68BL	30mm (C5)	27 mm (C5)	3 mm(C5) on 09-07-11 GED 3 m	3 mm(C5) on 09-07-11 GED 3 m	Natural fall occurred on 10-07-11 and on 18-07-11.
3.	68AL	32 mm (C5)	26 mm (C5)	3 mm(C5) on 09-07-11 GED 10 m	2 mm(C5) on 09-07-11 GED 10 m	Natural fall occurred on 10-07-11,18-07-11 & 26-07-11.
4.	68L	41mm (C5)	33mm (C5)	3 mm(C5) on 28-07-11 GED 6 m	3 mm(C5) on 28-07-11 GED 6m	Natural fall occurred on 18-07-11 and on 29-07-11.
5.	67BL	25 mm (C4)	22 mm (C4)	2.0 mm(C4) on 23-07-11 GED 4 m	2.0 mm(C4) on 23-07-11 GED 4 m	Natural fall occurred on 08-07-11.
6.	67AL	***34 mm (C3)***	***18 mm (C3)***	2 mm(C3) on 18-07-11 GED 8m	1 mm(C3) on 18-07-11 GED 8 m	No fall

7.	67L	31 mm (C2)	23mm (C2)	2 mm(C2) on 20-07-11 GED 3 m	2 mm(C2) on 20-07-11 GED 3 m	No fall
8.	66BL	18 mm (C1)	7mm (C1)	2 mm(C1) on 12-07-11 GED 6 m	1 mm(C2) on 12-07-11 GED 6 m	No fall
9.	66AL	10 mm (C1)	8mm (C1)	2 mm(C1) on 29-07-11 GED 6 m	1 mm(C1) on 29-07-11 GED 6 m	No fall

Table 5.3 Convergence Observation Up to the End Of The Month Of August –11

Sl No	**Location**	**Convergence observation**				
		Cumulative convergence		**Max. convergence change in a day**		**Stone fall details**
		(0.5m)	**(2.5m)**	**(0.5m)**	**(2.5m)**	
1.	70L	12mm (C8)	8mm (C8)	1mm(C8) on 24-08-2011 GED 3m	1mm(C8) on 23-08-2011 GED 3 m	Natural fall occurred on 10-08-11.
2.	68BL	21mm (C8)	11 mm (C8)	3 mm(C8) on 09-08-11 GED 12 m	2 mm(C8) on 09-08-11 GED 12 m	Natural fall occurred on 03-08-11, 10-08-11 and on 30-08-11.
3.	68AL	22 mm (C7)	20 mm (C7)	2 mm(C7) on 28-08-11 GED 4 m	2 mm(C7) on 28-08-11 GED 4 m	Natural fall occurred on 10-08-11.
4.	68L	**35mm (C6)**	**28mm (C6)**	3 mm(C6) on 30-08-11 GED 3 m	3 mm(C5) on 30-08-11 GED 3 m	Natural fall occurred on 18-07-11 and on 29-07-11.

5.	67BL	36 mm (C6)	21 mm (C6)	3 mm(C6) on 26-08-11 GED 4 m	3 mm(C6) on 26-08-11 GED 4 m	Natural fall occurred on 01-08-11, 03-08-11 and on 27-08-11.
6.	67AL	23 mm (C5)	14 mm (C5)	3 mm(C5) on 27-08-11 GED 3 m	3 mm(C5) on 21-08-11 GED 5 m	Natural fall occurred on 01-08-11, 03-08-11 and on 27-08-11.
7.	67L	24 mm (C5)	22mm (C5)	3 mm(C5) on 30-08-11 GED 10 m	3 mm(C5) on 30-08-11 GED 10 m	Natural fall occurred on 12-08-11 and on 30-08-11.
8.	66BL	34 mm (C3)	30mm (C3)	2 mm(C3) on 27-08-11 GED 3 m	3 mm(C3) on 27-08-11 GED 3 m	Natural fall occurred on 17-08-11 and on 30-08-11.
9.	66AL	12 mm (C3)	11 mm (C3)	2 mm(C3) on 29-08-11 GED 10 m	2 mm(C3) on 29-08-11 GED 10 m	No fall

Table 5.4 Convergence Observation Up to the End Of The Month Of September –11

Sl No	**Location**	**Convergence observation**				
		Cumulative convergence		**Max. convergence change in a day**		**Stone fall details**
		(0.5m)	**(2.5m)**	**(0.5m)**	**(2.5m)**	
1	68BL	7 (C10)	8 (C10)	2mm on 3,10/9/11 GED 17,11m respectively	2mm on 2,8/9/11 GED 17,13m respectively	No fall

2.	68AL	5 (C9)	5 (C9)	1mm on 4,5,12/9/11 GED 14,12,6m respectively	1mm on 4,5,12/9/11 GED 14, 12m respectively	Natural fall occurred on 07-09-11
3.	68L	15 (C8)	12 (C8)	1mm on 11,12,29/9/11 GED 16,12,10m respectively	1mm on 18,19,30/9/11 GED 16,16,10m respectively	Natural fall occurred on 01,07-09-11
4.	67BL	23 (C7)	19 (C7)	2mm on 4,13/9/11 GED 10,6m respectively	2mm on 2, 15/9/11 GED 6, 12m respectively	No Fall
5.	67L	**47** **(C5)**	**41** **(C5)**	1mm on 23,24/9/12 GED 4m	1mm on 21,24/9/12 GED 4m	No fall
6	66BL	30 (C4)	28 (C4)	2mm on 4,5,6/9/11 GED 6,4,4 m respectively	2mm on 2,7/9/11 GED 8,4m respectively	No fall
7	66AL	26 (C4)	21 (C4)	3mm on 9/9/11 GED 10M	2mm on 12/9/11 GED 8m	No fall

Table 5.5: Convergence Observation Up to the End of the Month of October –11

Sl No	**Location**	**Convergence observation**				**Stone fall details**
		Cumulative convergence		**Max. convergence change in a day**		
		(0.5m)	**(2.5m)**	**(0.5m)**	**(2.5m)**	
1	68L	30 (C8)	25 (C8)	1mm on 12,19,24/10/12 GED 12m	1mm on 12,18,24/10/12 GED 12,12,10m	Natural fall occurred on 28,31-10-11

2	67BL	28 (C8)	24 (C8)	1mm on 5, 9,23,24/10/12 GED 14m	1mm on 5,9,15,29/10/12 GED 14m	Natural fall occurred on 27-10-11
3	67AL	32 (C6)	28 (C6)	2mm on 22/10/12 GED 16m	2mm on 22/10/12 GED 16m	No fall
4.	67L	**51 (C6)**	**40 (C6)**	1mm on 21/10/12 GED 4m	1mm on 21/10/12 GED 4m	Natural fall occurred on 27-10-11
5.	66BL	33 (C5)	29 (C5)	2mm on 25/10/12 GED 2m	2mm on 22/10/12 GED 8m	No fall
6.	66AL	14 (C5)	11 (C5)	2mm on 22/10/12 GED 25m	2mm on 27/10/12 GED 23m	No fall

Table 5.6: Convergence Observation Up to the End of the Month Of November –11

Sl No	**Location**	**Convergence observation**				**Stone fall details**
		Cumulative convergence		**Max. convergence change in a day**		
		(0.5m)	**(2.5m)**	**(0.5m)**	**(2.5m)**	
1	68L	27 (C9)	24 (C9)	3mm on 10, 11/11/11 GED 10,8m	3mm on 10, 11/11/11 GED 10,8m	Natural fall occurred on 5,13,23-11-11
2	67BL	10 (C9)	10 (C9)	2mm on 11,12/11/11 GED 7, 5m	2mm on 12,13/11/11 GED 5m	No fall
3	67AL	**19 (C9)**	**9 (C9)**	2mm on 12, 20/11/11 GED 12, 43m	2mm on 10, 11, 13, 20/11/11 GED 13, 12, 10,4m	Natural fall occurred on 13-11-11

4.	67L	28 (C9)	25 (C9)	3mm on 24, 26, 28 /11/11 GED 12, 12, 9m	3mm on 20,24/11/11 GED 12,17m	Natural fall occurred on 28-11-11
5.	66BL	17 (C7)	16 (C7)	3mm on 11/11/11 GED 8m	3mm on 13/11/11 GED 5m	Natural fall occurred on 3-11-11
6.	66AL	38 (C4)	30 (C4)	3mm on 18, 20, 24, 27/11/11 GED 15, 13, 8& 4m respectively	3mm on 18/11/11 GED 15m	Natural fall occurred on 3,23-11-11

Table 5.7: Convergence Observation Up to the End of the Month of December –11

Sl. No.	**Location**	**Convergence observation**				
		Cumulative convergence		**Max. convergence change in a day**		**Stone fall details**
		(0.5m)	**(2.5m)**	**(0.5m)**	**(2.5m)**	
1	67BL	33 (C10)	30 (C10)	2mm on 1,2/12//11 GED 5m respectively	2mm on 1, 7/12//11 GED 5m respectively	No fall
2	67AL	25 (C11)	24 (C11)	3mm on 2, 10/12//11 GED 7, 5m respectively	2mm on 3, 12/12//11 GED 7, 5m respectively	No fall
3	67L	36 (C11)	33 (C11)	4mm on 17/12//11 GED5m	3mm on 17/12//11 GED 5m	No fall

4	66BL	**38 (C10)**	**34 (C10)**	3mm on 4, 17/12//11 GED 14, 7m respectively	2mm on 14,15/12/11 GED 8m	No fall
5	66AL	36 (C9)	35 (C9)	3mm on 17/12//11 GED 15M	3mm on 17/12/11 GED 15m	Natural fall occurred on 10,23-12-11

Table 5.8 Convergence Observation Up to the End of the Month of January–12

Sl No	**Location**	**Convergence observation**				
		Cumulative convergence		**Max. convergence change in a day**		**Stone fall details**
		(0.5m)	**(2.5m)**	**(0.5m)**	**(2.5m)**	
1.	66AL	15 (C10)	13 (C10)	2mm on 2, 3/1//12 GED 8M	2mm on 3/1/12 GED 8m	No fall

Before the occurrence of main fall on 2nd September, 2011, rate of convergence at a distance of about 10, 15m and 20 m was 2-3 mm/day for five days, 1 mm/ day for four days, 1 mm/ day for three days, respectively. Goaf Edge Distance (GED) was about 21 m at the time of installation of the convergence station C-10 in 66 B Level. Maximum convergence of 38 mm was recorded when the station reached goaf edge i.e at a distance of about 3 m from the goaf edge, beyond which the monitoring of convergence with manually was not possible.

Graphical Representation of the daily Convergence observations at different convergence stations in different levels like 70, 68, 68A, 68B, 67, 67A, 67B, 66, 66A, 66B is shown below in the figures 5.1-5.9.

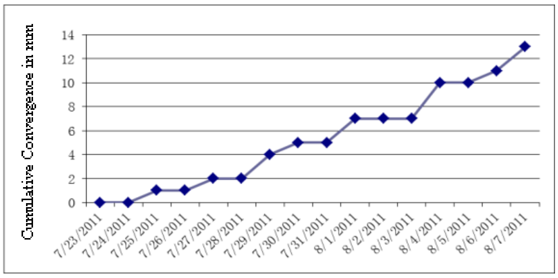


Fig. 5.1.1 Convergence Station C 1

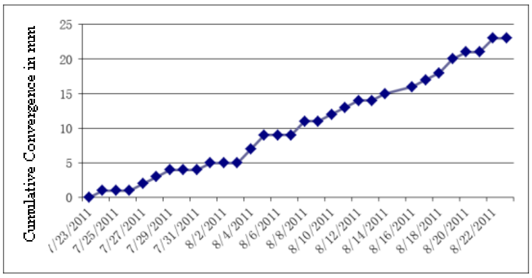


Fig. 5.1.2 Convergence Station C 2

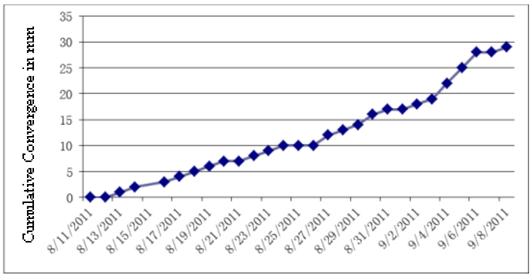


Fig. 5.1.3 Convergence Station C 3

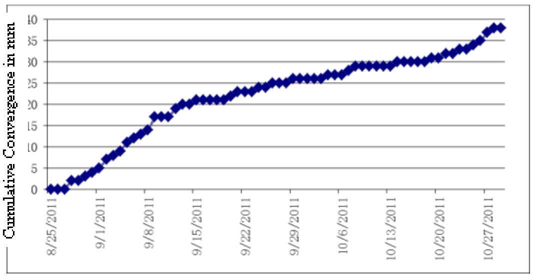


Fig. 5.1.4 Convergence Station C 4

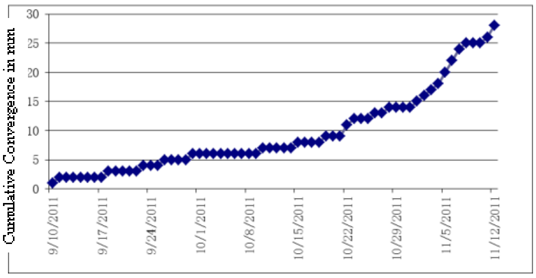


Fig. 5.1.5 Convergence Station C 5

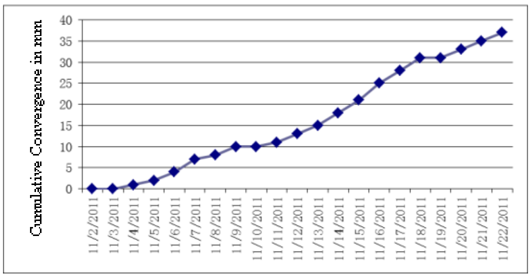


Fig. 5.1.6 Convergence Station C 6

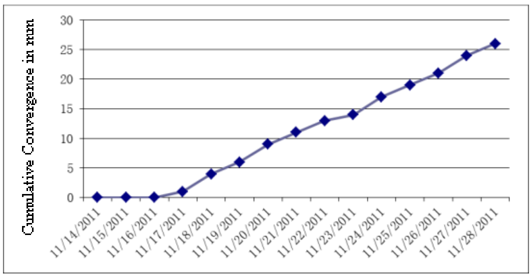


Fig. 5.1.7 Convergence Station C 7

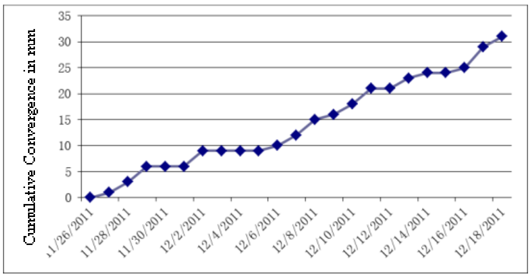


Fig. 5.1.8 Convergence Station C 8

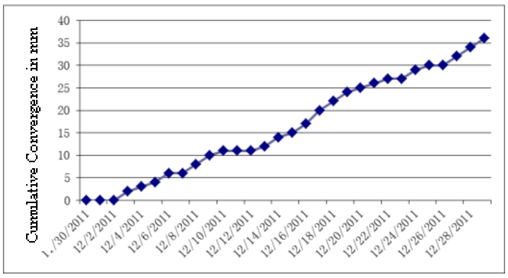


Fig. 5.1.9 Convergence Station C 9

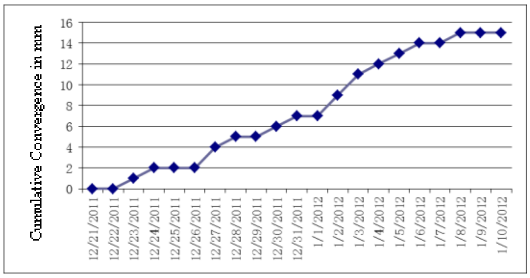


Fig. 5.1.10 Convergence Station C 10

Fig.5.1 Convergence observations in level 66A

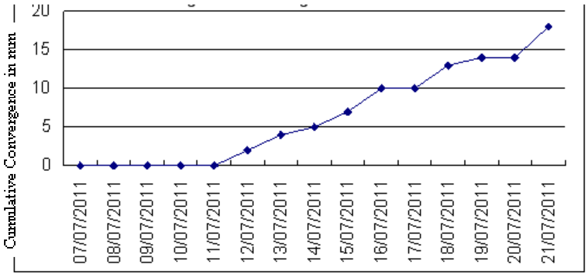


Fig. 5.2.1 Convergence Station C 1

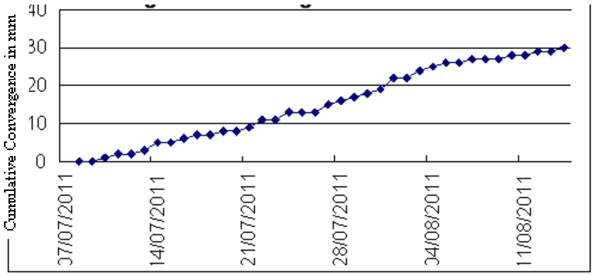


Fig. 5.2.2 Convergence Station C 2

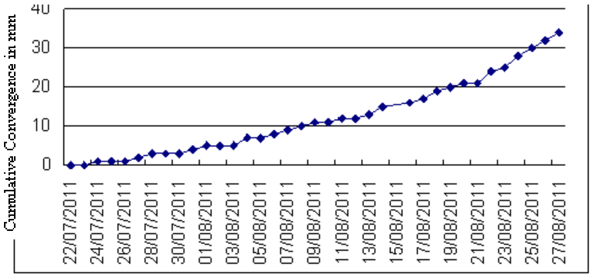


Fig. 5.2.3 Convergence Station C 3

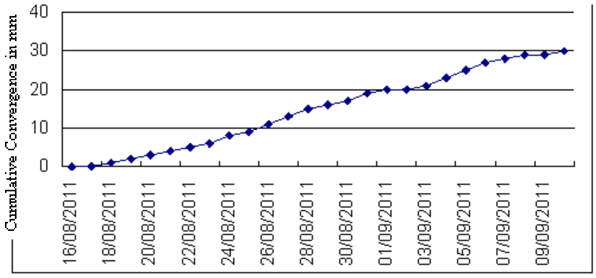


Fig. 5.2.4 Convergence Station C 4

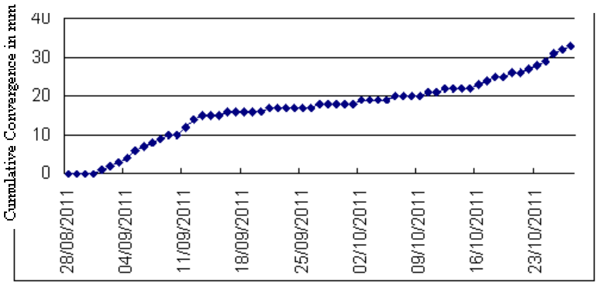


Fig. 5.2.5 Convergence Station C 5

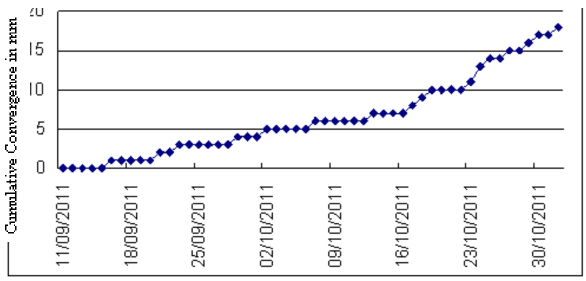


Fig. 5.2.6 Convergence Station C 6

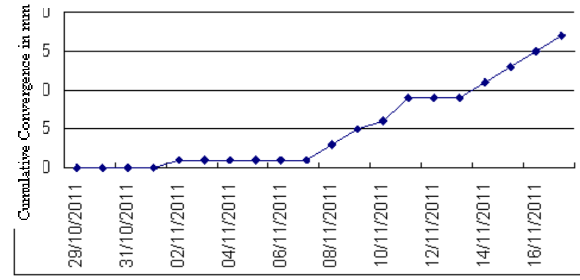


Fig. 5.2.7 Convergence Station C 7

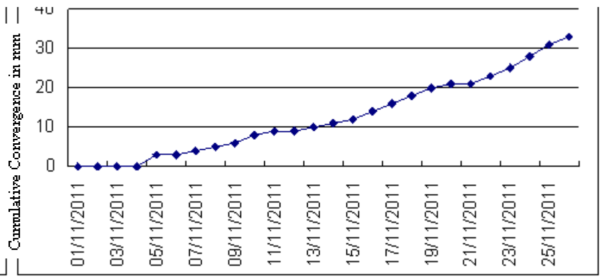


Fig. 5.2.8 Convergence Station C 8

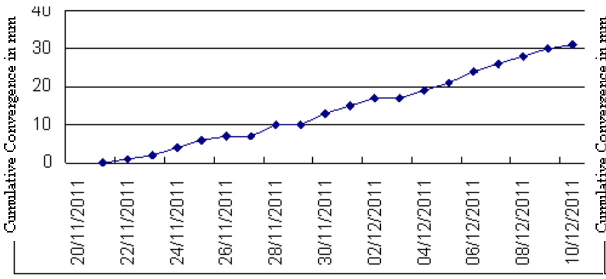


Fig. 5.2.9 Convergence Station C 9

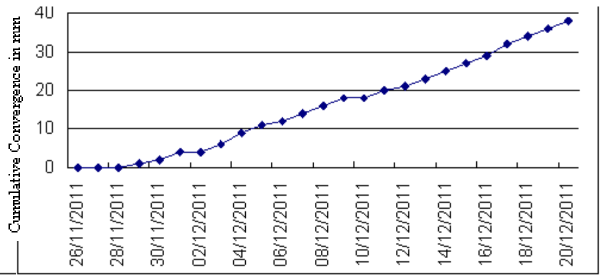


Fig. 5.2.10 Convergence Station C 10

Fig.5.2 Convergence observations in Level 66B

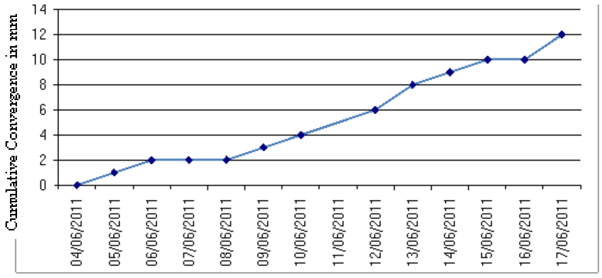


Fig. 5.3.1 Convergence Station C 1

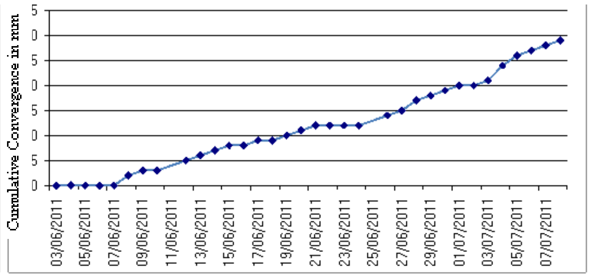


Fig. 5.3.2 Convergence Station C 2

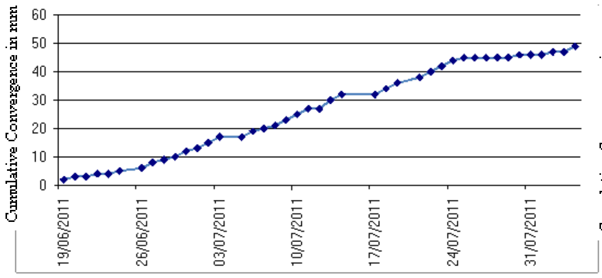


Fig. 5.3.3 Convergence Station C 3

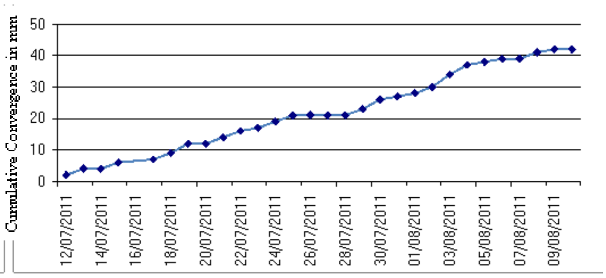


Fig. 5.3.4 Convergence Station C 4

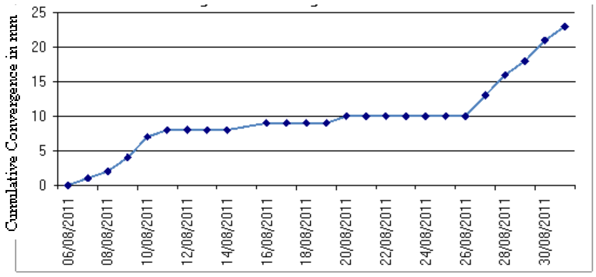


Fig. 5.3.5 Convergence Station C 5

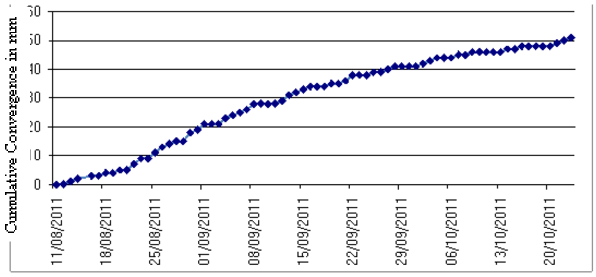


Fig. 5.3.6 Convergence Station C 6

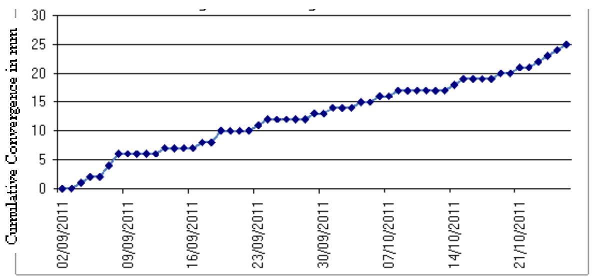


Fig. 5.3.7 Convergence Station C 7

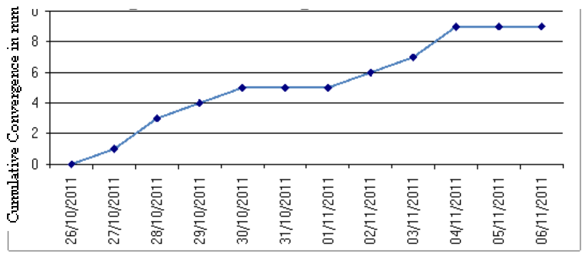


Fig. 5.3.8Convergence Station C 8

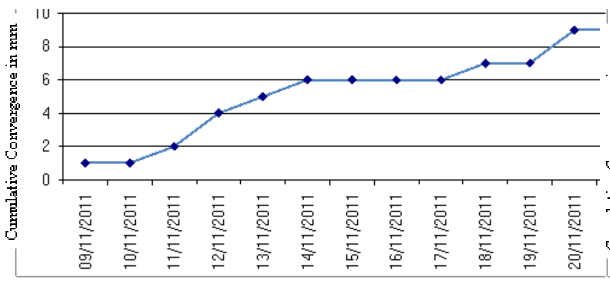


Fig. 5.3.9 Convergence Station C 9

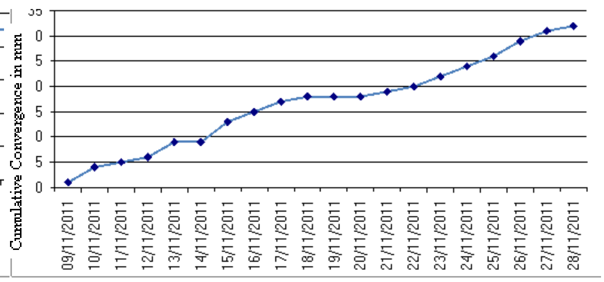


Fig. 5.3.10 Convergence Station C 10

Fig. 5.3 Convergence observations in Level 67A

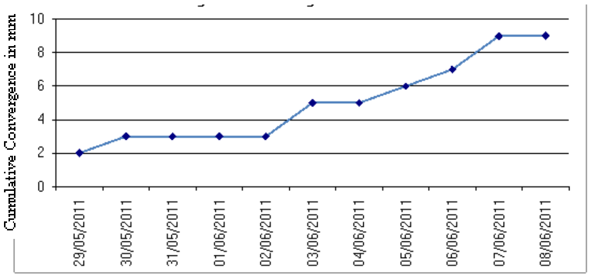


Fig. 5.4.1 Convergence Station C 1

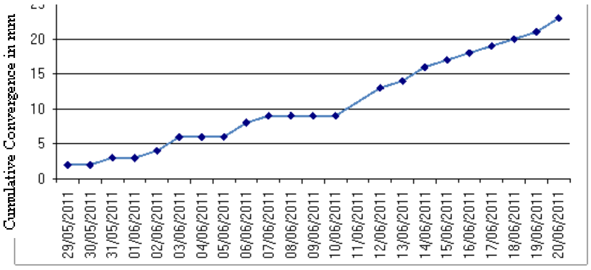


Fig. 5.4.2 Convergence Station C 2

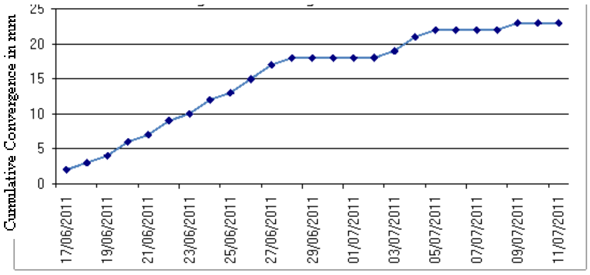


Fig. 5.4.3 Convergence Station C 3

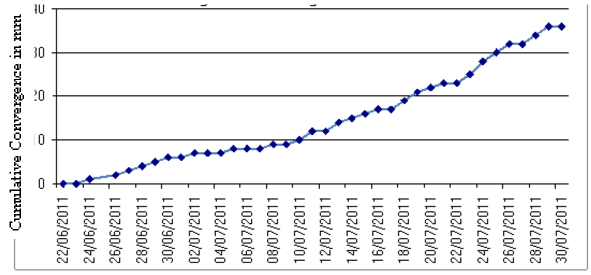


Fig. 5.4.4 Convergence Station C 4

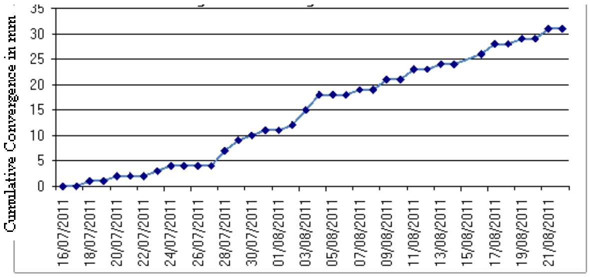


Fig. 5.4.5 Convergence Station C 5

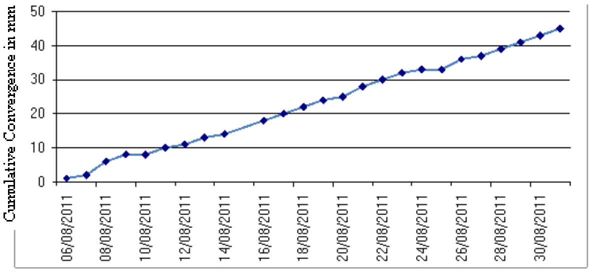


Fig. 5.4.6 Convergence Station C 6

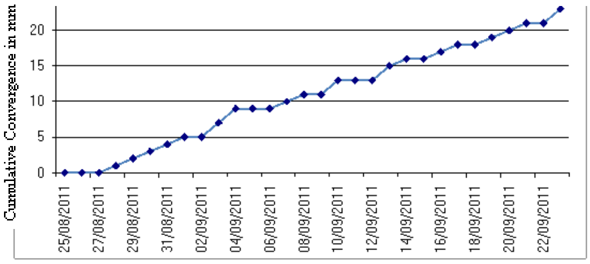


Fig. 5.4.7 Convergence Station C 7

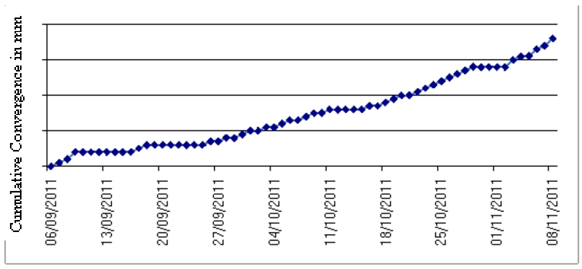


Fig. 5.4.8 Convergence Station C 8

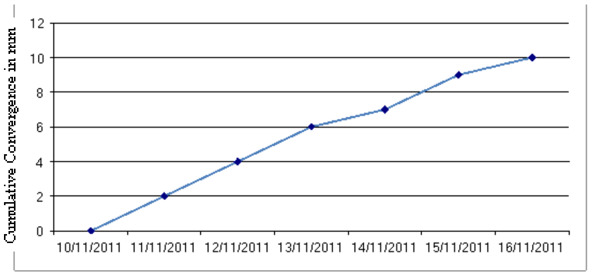


Fig. 5.4.9 Convergence Station C 9

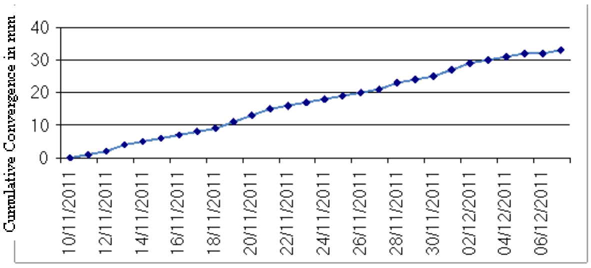


Fig. 5.4.10 Convergence Station C 10

Fig. 5.4 Convergence Observations in Level 67 B

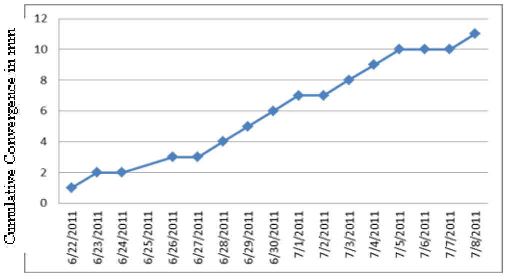


Fig. 5.5.1 Convergence Station C 1

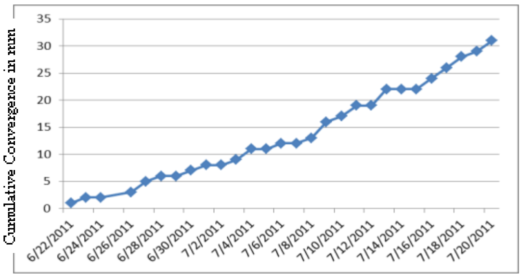


Fig. 5.5.2 Convergence Station C 2

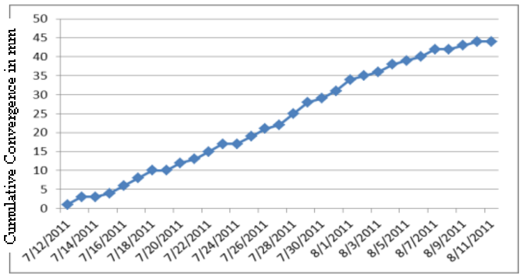


Fig. 5.5.3 Convergence Station C 3

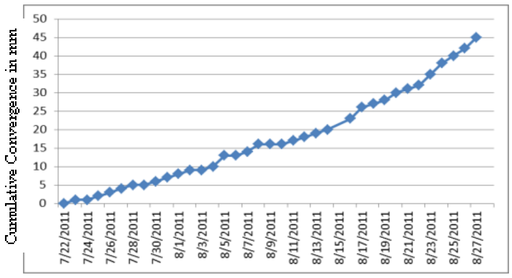


Fig. 5.5.4 Convergence Station C 4

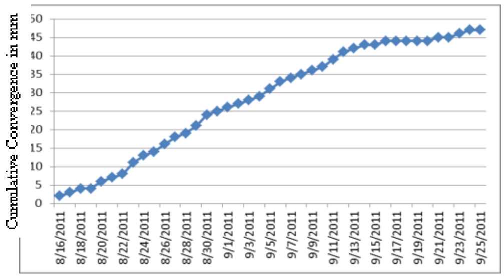


Fig. 5.5.5 Convergence Station C 5

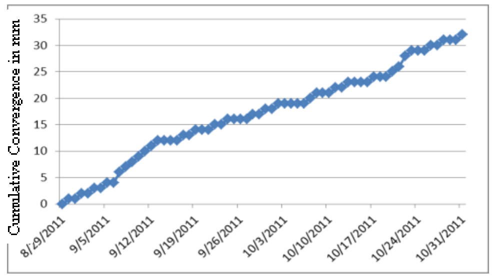


Fig. 5.5.6 Convergence Station C 6

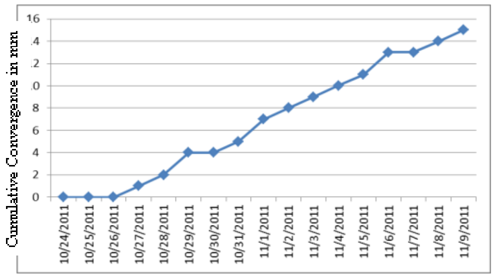


Fig. 5.5.7 Convergence Station C 7

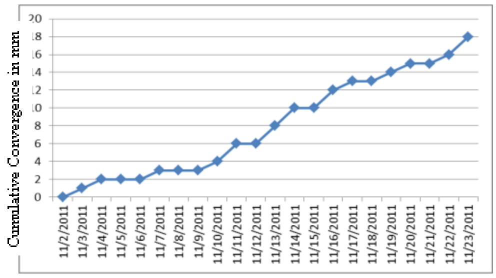


Fig. 5.5.8 Convergence Station C 8

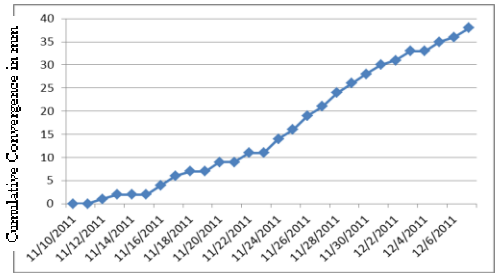


Fig. 5.5.9 Convergence Station C 9

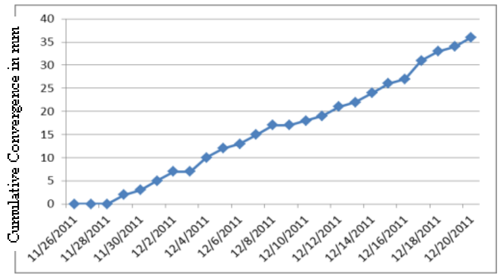


Fig. 5.5.10 Convergence Station C 10

Fig.5.5Convergence observations in Level 67

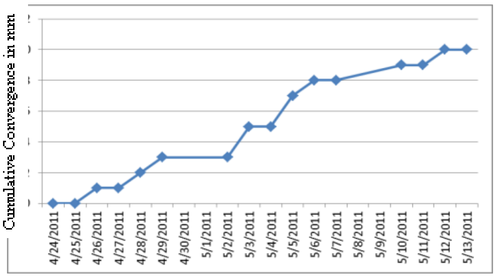


Fig. 5.6.1 Convergence Station C 1

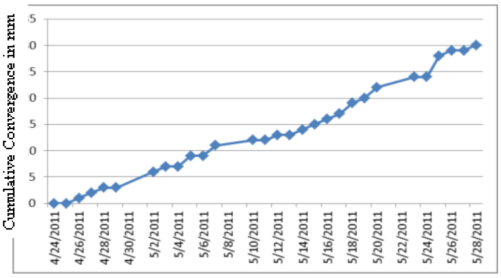


Fig. 5.6.2 Convergence Station C 2

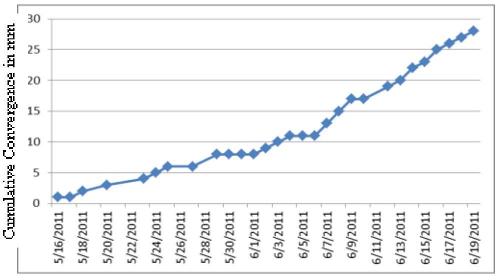


Fig. 5.6.3 Convergence Station C 3

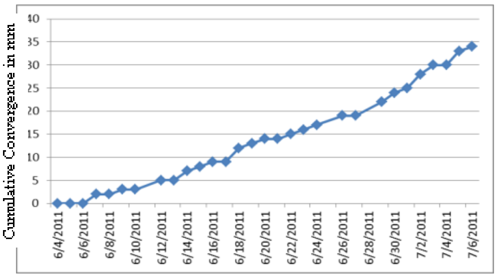


Fig. 5.6.4 Convergence Station C 4

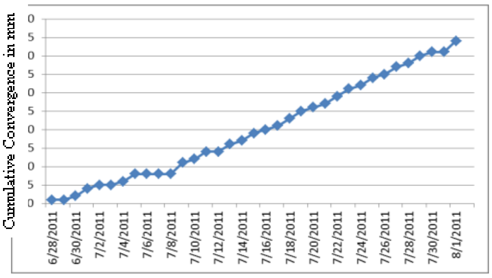


Fig. 5.6.5 Convergence Station C 5

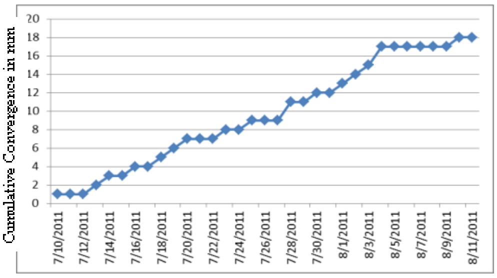


Fig. 5.6.6 Convergence Station C 6

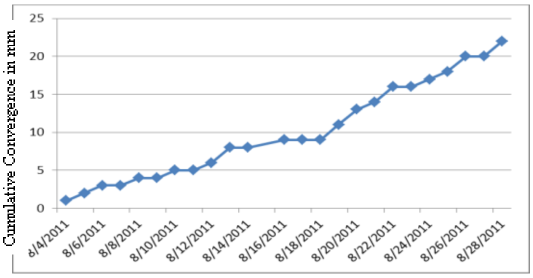


Fig. 5.6.7 Convergence Station C 7

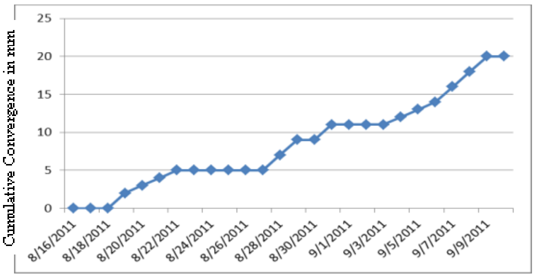


Fig. 5.6.8 Convergence Station C 8

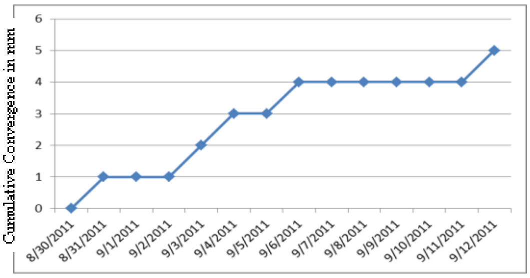


Fig. 5.6.9 Convergence Station C 9

Fig.5.6 Convergence observations in Level 68A

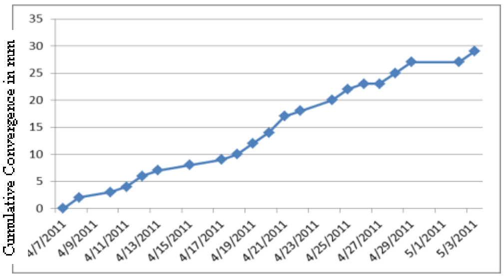


Fig. 5.7.1 Convergence Station C 1

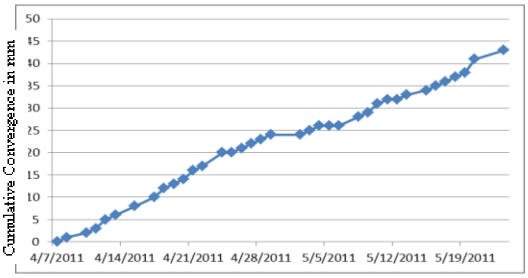


Fig. 5.7.2 Convergence Station C 2

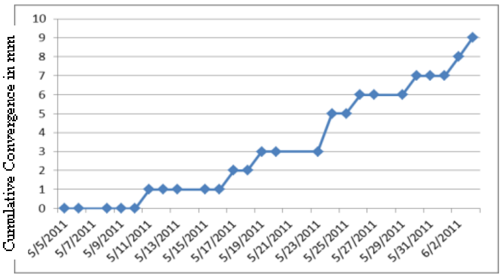


Fig. 5.7.3 Convergence Station C 3

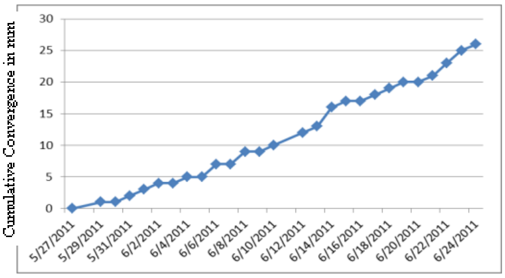


Fig. 5.7.4 Convergence Station C 4

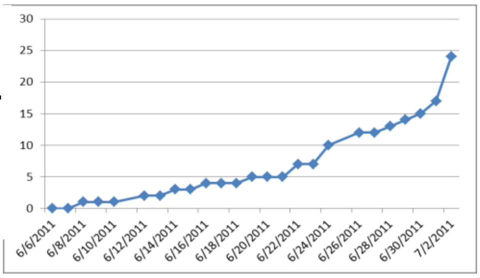


Fig. 5.7.5 Convergence Station C 5

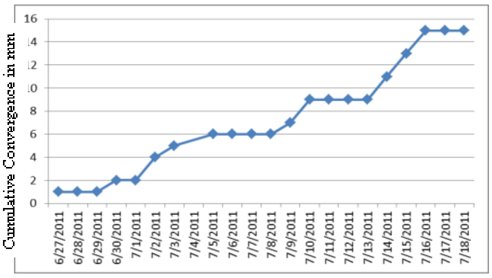


Fig. 5.7.6 Convergence Station C 6

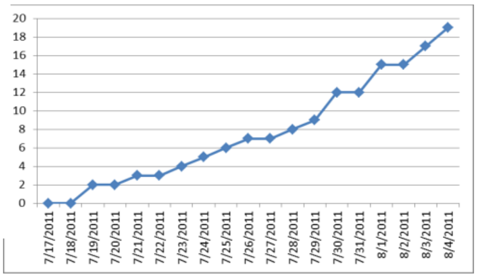


Fig. 5.7.7 Convergence Station C 7

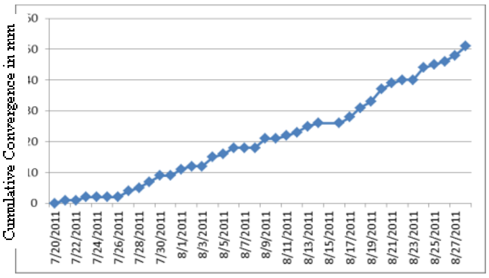


Fig. 5.7.8 Convergence Station C 8

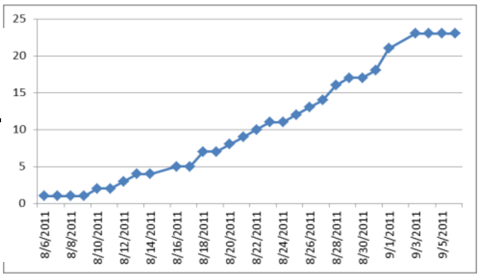


Fig. 5.7.9 Convergence Station C 9

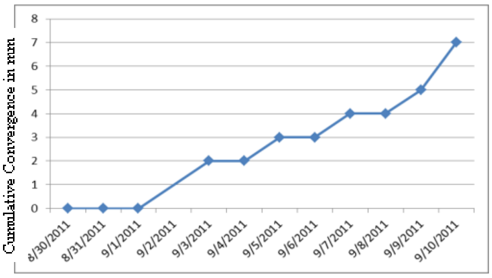


Fig. 5.7.10Convergence Station C 10

Fig.5.7 Convergence observations in Level 68B

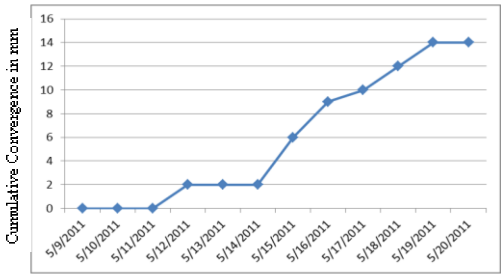


Fig. 5.8.1 Convergence Station C 1

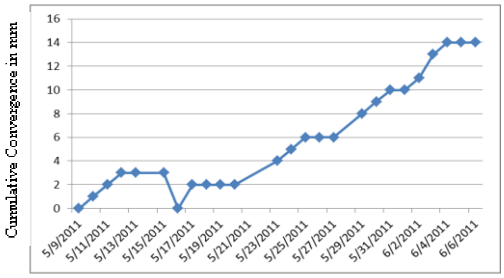


Fig. 5.8.2 Convergence Station C 2

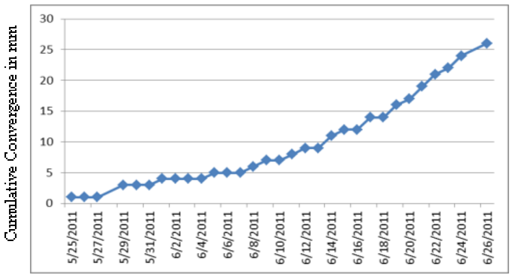
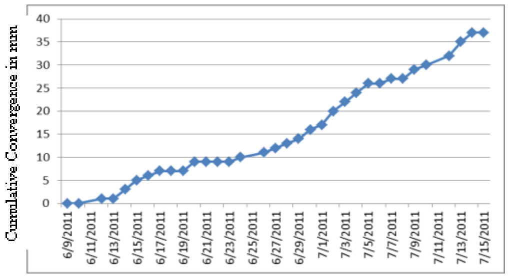


Fig. 5.8.3 Convergence Station C 3



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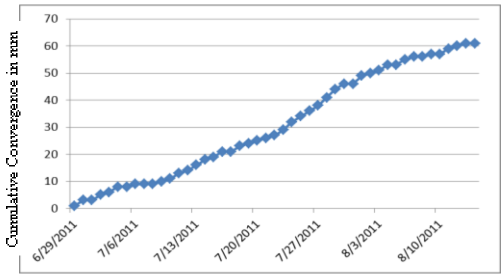

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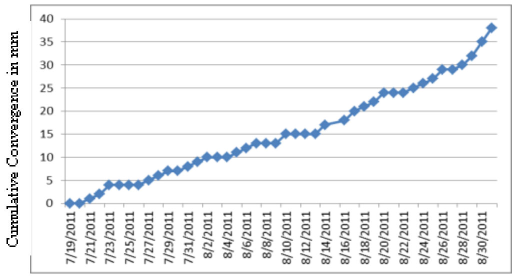


Fig. 5.8.6 Convergence Station C 6

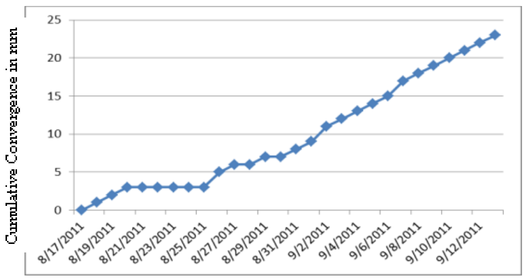


Fig. 5.8.7 Convergence Station C 7

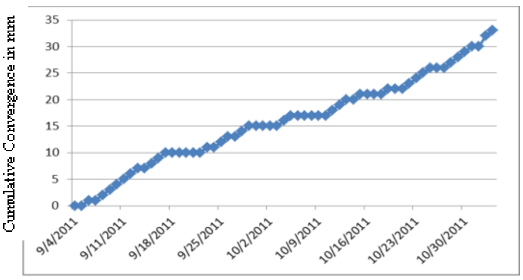


Fig. 5.8.8 Convergence Station C 8

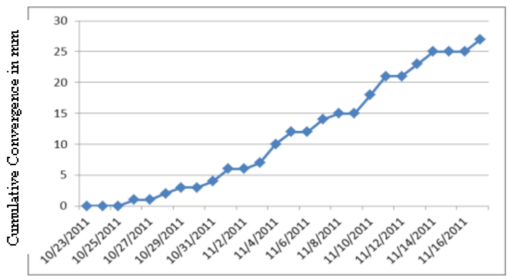


Fig. 5.8.9 Convergence Station C 9

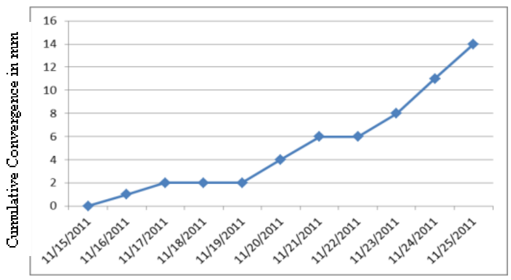
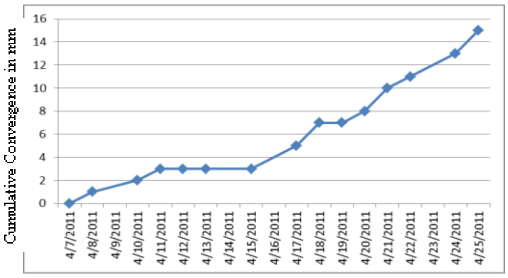


Fig. 5.8.10 Convergence Station C 10

Fig. 5.8 Convergence observations in Level 68



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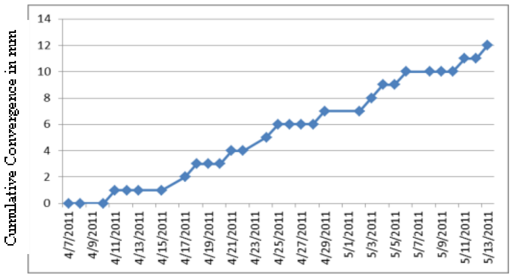

Fig. 5.9.2 Convergence Station C 2

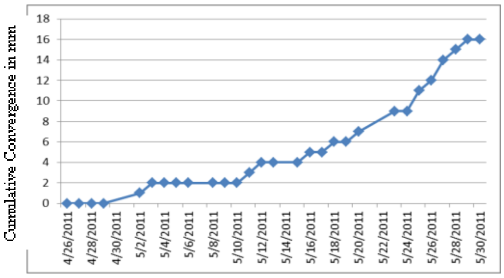

Fig. 5.9.3 Convergence Station C 3

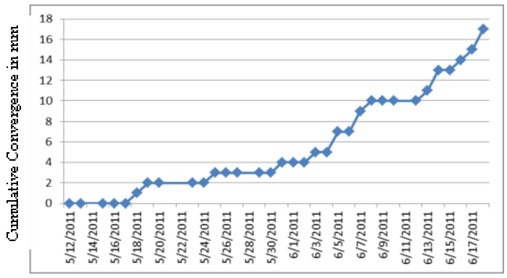

Fig. 5.9.4 Convergence Station C 4

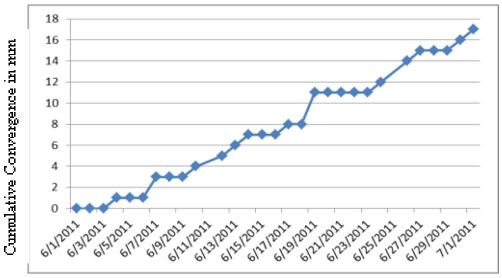

Fig. 5.9.5 Convergence Station C 5

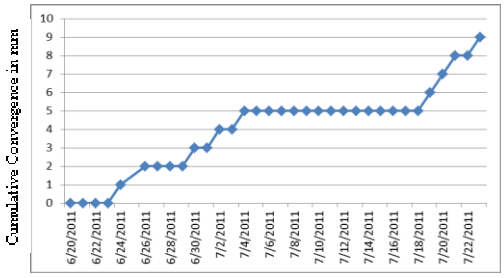

Fig. 5.9.6 Convergence Station C 6

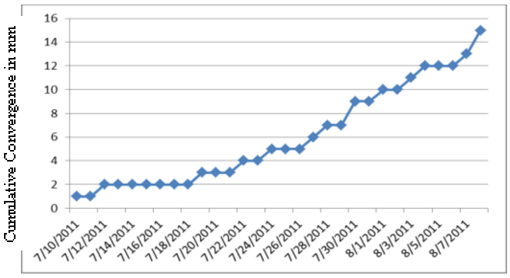

Fig. 5.9.7 Convergence Station C 7

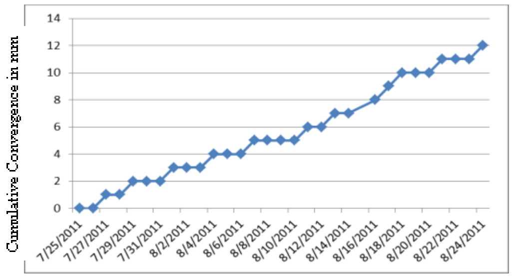

Fig. 5.9.8 Convergence Station C 8

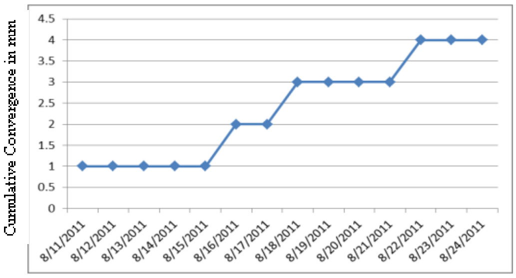

Fig. 5.9.9 Convergence Station C 9

Fig. 5.9 Convergence observations in 70L

CHAPTER 6

RESULTS

&

DISCUSSIONS

6. RESULTS

Extensive application of numerical modeling was done for understanding the stability of workings for extraction of pillars in thick coal seams. Depillaring process in the BG panel includes different stages of division of pillars into stooks and extraction of stooks with ring drilling or blasting up to full seam thickness. For two dimensional representation of full seam extraction in 11m thick seam, vertical section with four galleries in an idealized panel was selected. A few parameters were kept constant for the models, e.g. width of pillar, development gallery and split gallery as 50 m, 4.2 m, and 4.2 m, respectively.

The problem domain consists of appropriate boundary conditions and grid pattern with development of three pillars of 50 m and 4.2 m wide galleries. The models simulated pillar extraction in plain strain conditions with Mohr Coulomb material. The model has its outer boundary located 150 m away from the mine panel. The top is free to move in any direction, but the bottom edge of the model is restricted from moving vertically and horizontally. Roller type boundary conditions for all the models were placed along two edges of the models. 204 models were simulated by using the most sophisticated software of geo-technics – FLAC, so as to calibrate to the field conditions.

6.1 Numerical Modelling Outputs

The roof deformation with respect to goaf edge distance is considered to be the criterion of the study. Hence the roof deformation in openings like galleries, splits, etc. during various stages of extraction is shown in the below figures. The maximum deformation is recognized by the coloured zone and its corresponding values displayed in meters.

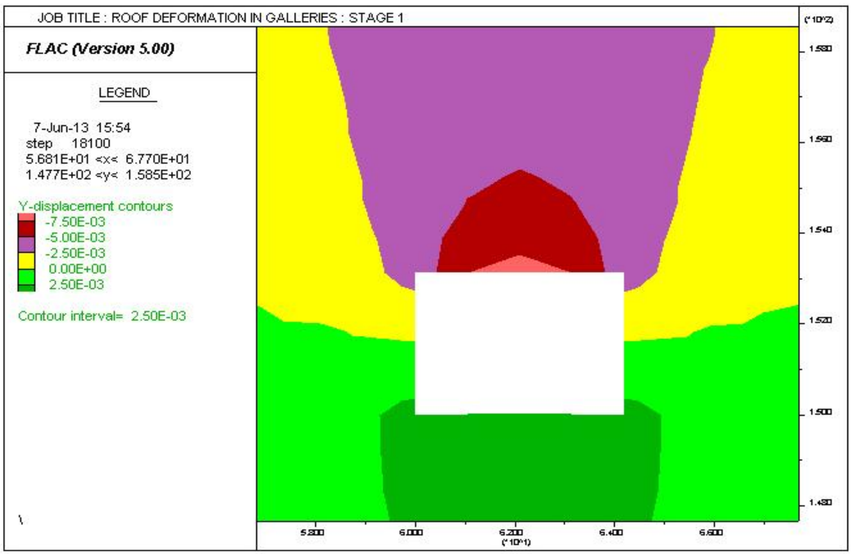

Fig. 6.1 Maximum Deformation of Roof after Development of Pillars

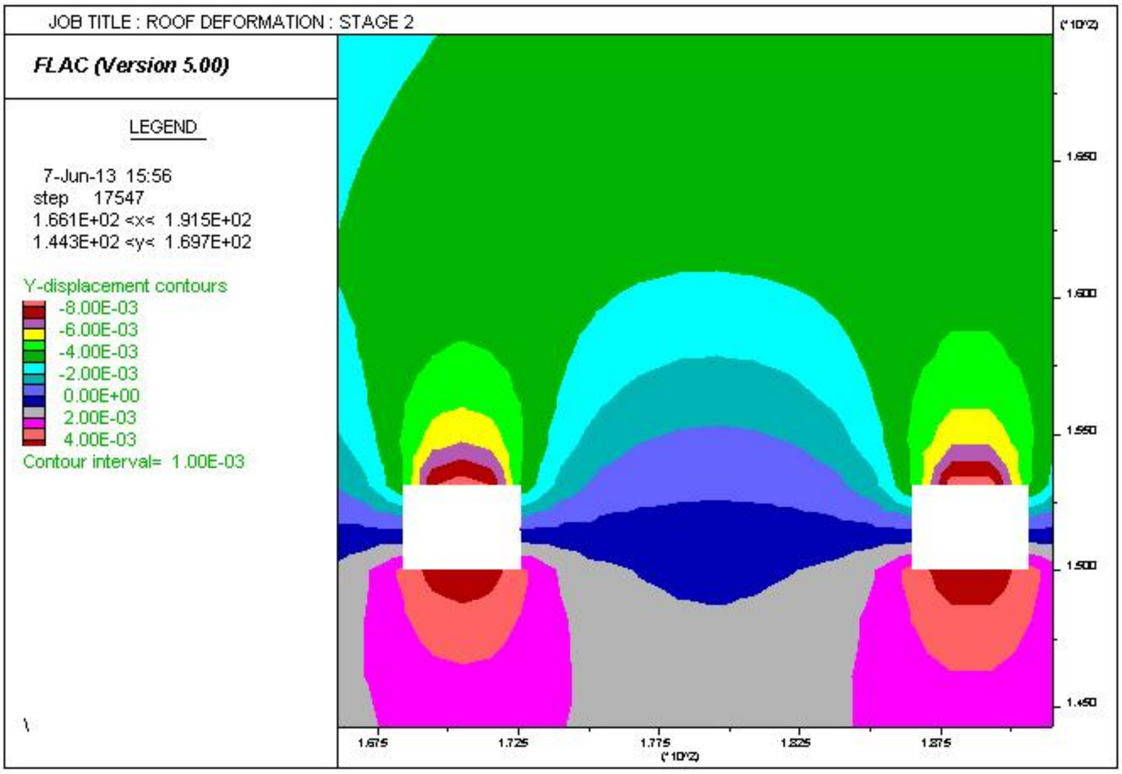

Fig. 6.2 Maximum Deformation of Roof after Splitting of Pillars

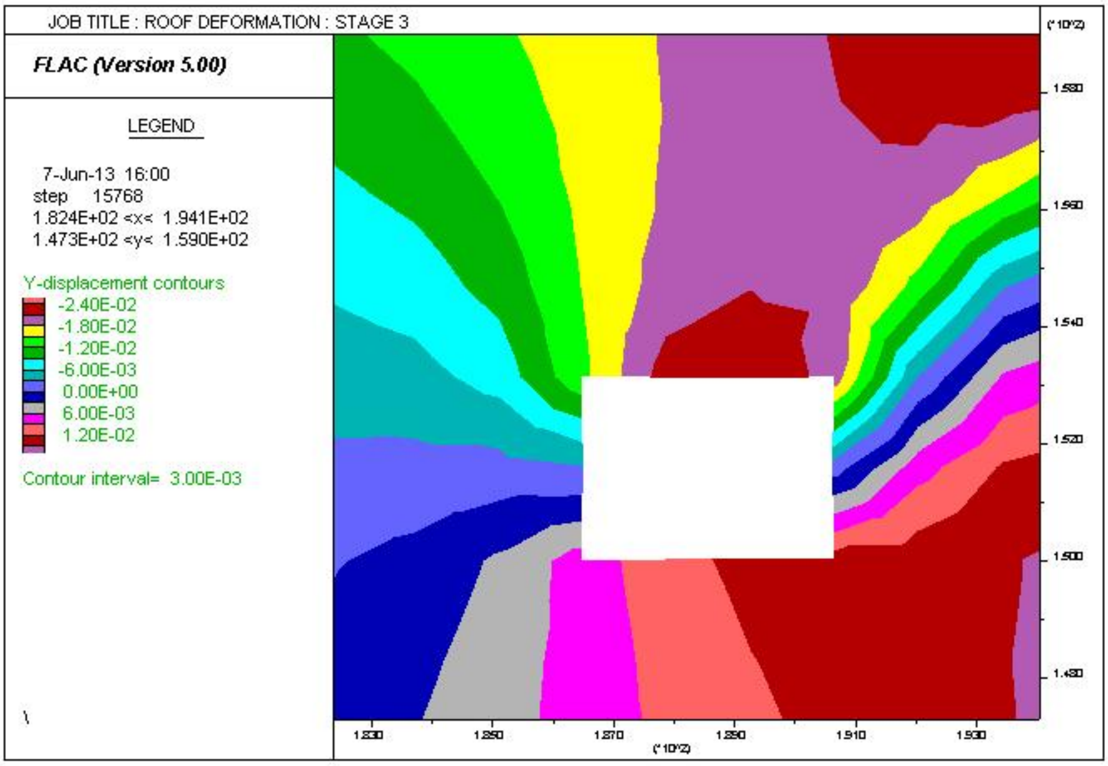

Fig. 6.3 Maximum Deformation of Roof after Extraction of 1 Stook

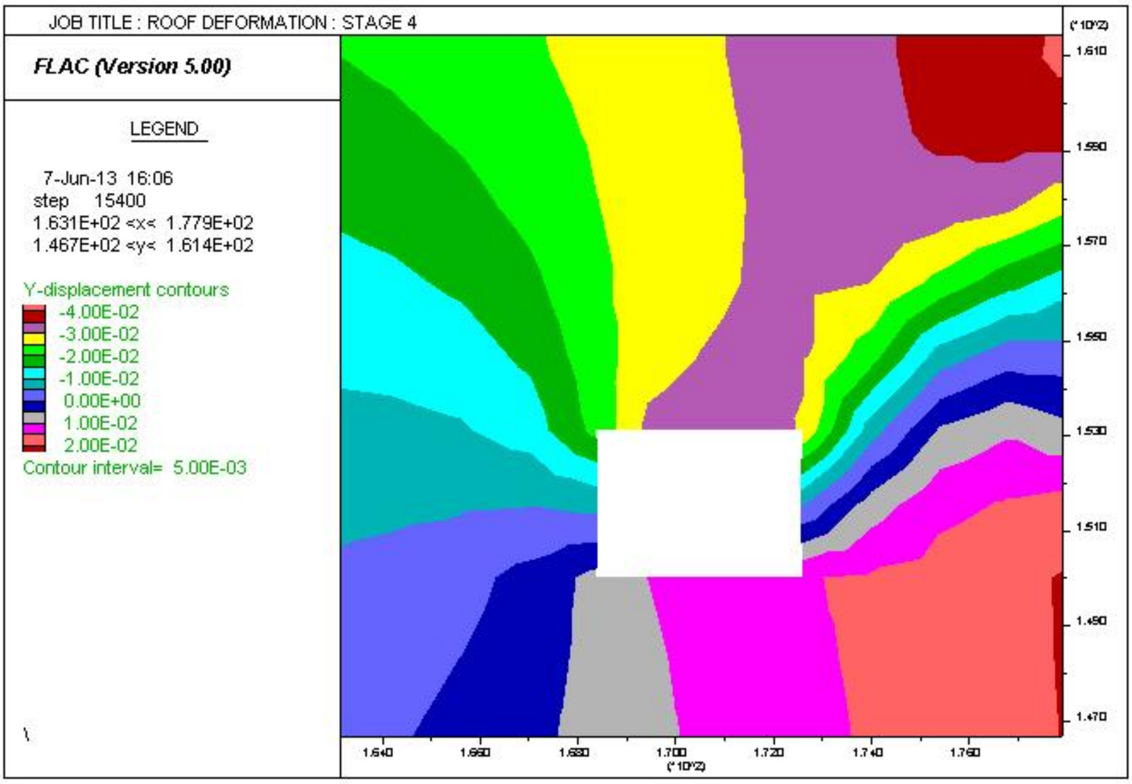

Fig. 6.4 Maximum Deformation of Roof after Extraction of 2 Stooks

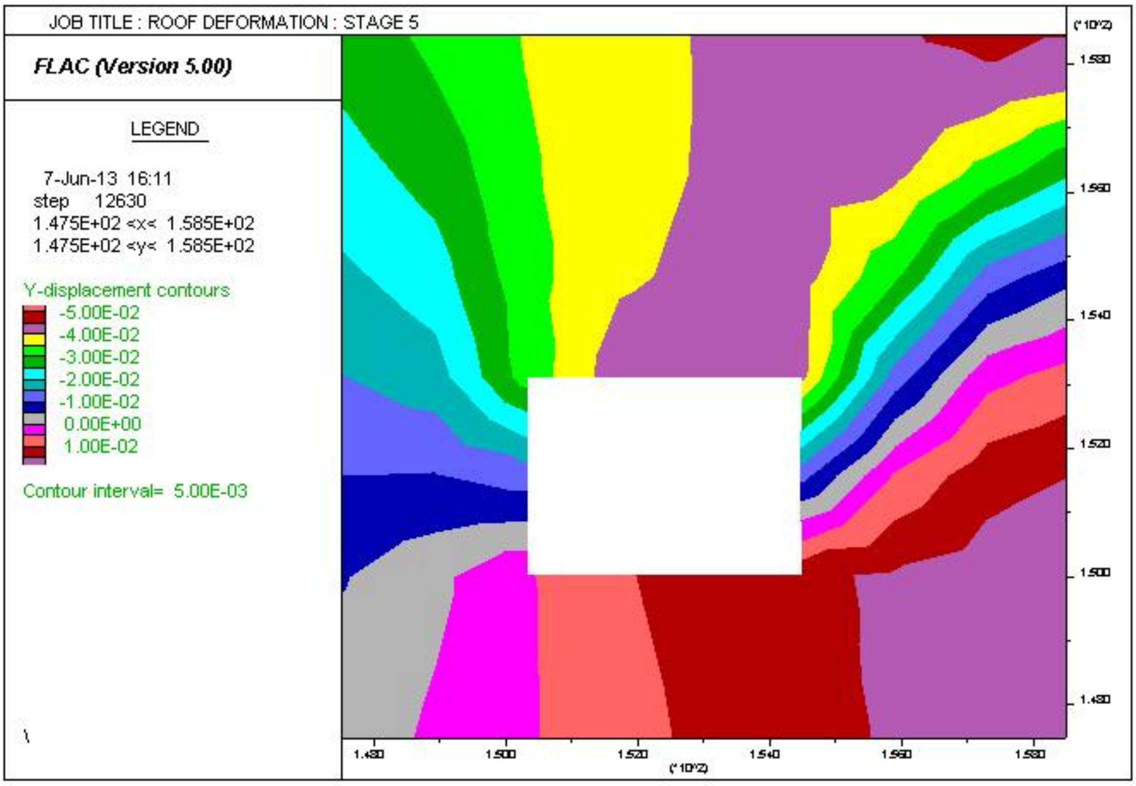

Fig. 6.5 Maximum Deformation of Roof after Extraction of 3 Stooks

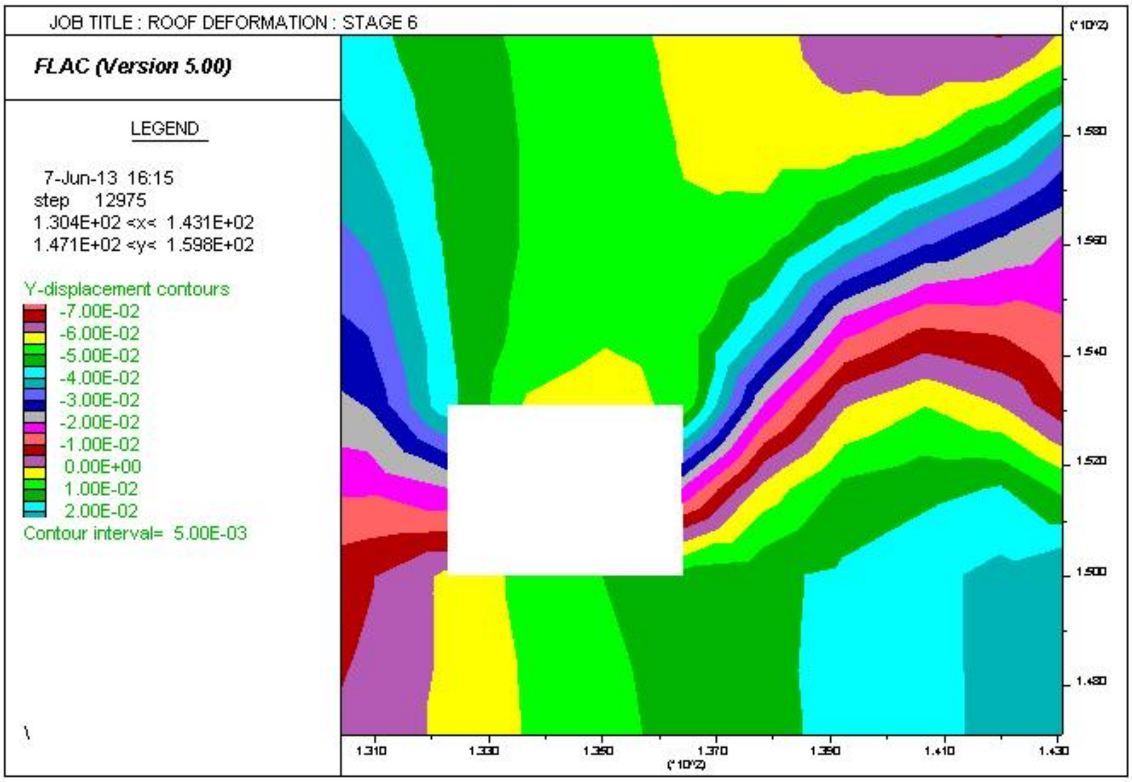

Fig. 6.6 Maximum Deformation of Roof after Extraction of 4 Stooks

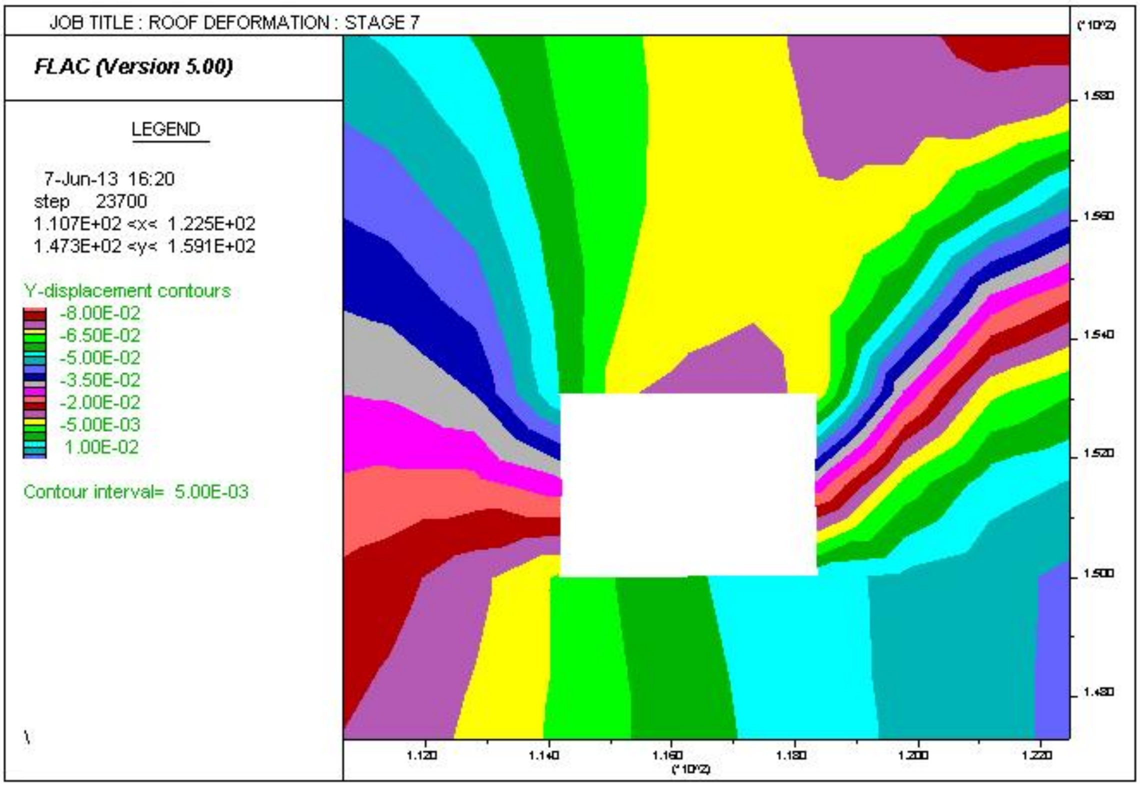

Fig. 6.7 Maximum Deformation of Roof after Extraction of 5 Stooks

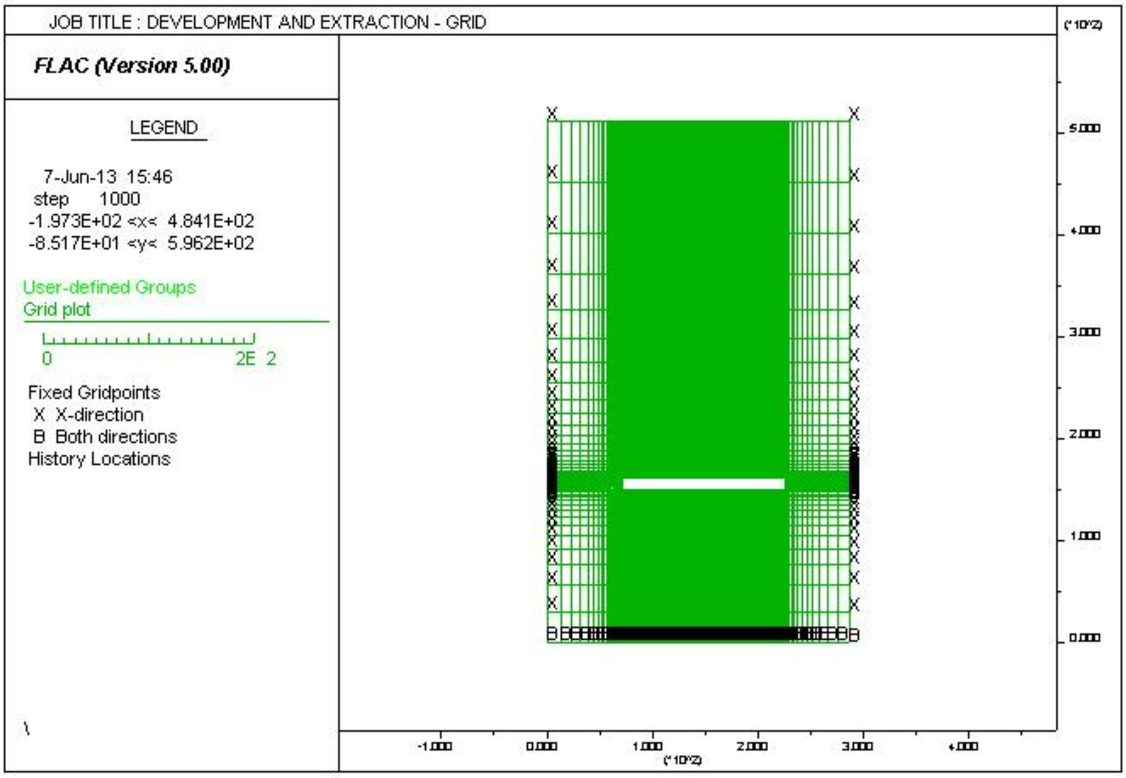

Fig. 6.8 Grid Generation of Final Stage after Extraction of 8 Stooks

The predicted values of the roof deformation is shown in the below Table 6.1

Table 6.1 Maximum deformation of roof in advance workings (Galleries and Splits) for various stages of extraction in BG panel in the numerical models

Stage of Extraction		**Location (Deformation in mm)**									
		G1	**S1**	**S2**	**G2**	**S3**	**S4**	**G3**	**S5**	**S6**	**G4**
I	Development	7.5	-	-	7.5	-	-	7.5	-	-	7.5
II	2 Splits	8	8	8	8	-	-	8	-	-	8
III	4splits,1stook extraction	-	-	21	12	12	12	12	-	-	12
IV	6splits,2 stooks extraction	-	-	-	30	25	15	15	15	15	15
V	3 stooks extraction	-	-	-	-	40	25	25	20	20	20
VI	4 stooks extraction	-	-	-	-	-	55	35	30	30	30
VII	5 stooks extraction	-	-	-	-	-	-	70	45	40	40

*G – Gallery, **S – Split

CHAPTER 7

VALIDATION OF MODEL

7. VALIDATION OF MODEL

7.1 Comparison of Modeling Results with Field Investigation Data

The Numerical modeling results were compared with that of the field observations. The various stages of extraction of thick coal seam was taken on X axis corresponding to the Goaf Edge Distance and Cumulative Convergence was taken on Y axis in figure 7.1. The modeling results and field observations indicated a maximum cumulative convergence of 70mm and 61mm respectively.

The different stages in the extraction of the coal block by Blasting Gallery panels in GDK 10 Incline include:

- Stage I: Development of galleries
- Stage II: After splitting
- Stage III: After extraction of one stook
- Stage IV: After extraction of two stooks
- Stage V: After extraction of three stooks
- Stage VI: After extraction of four stooks
- Stage VII: After extraction of five stooks

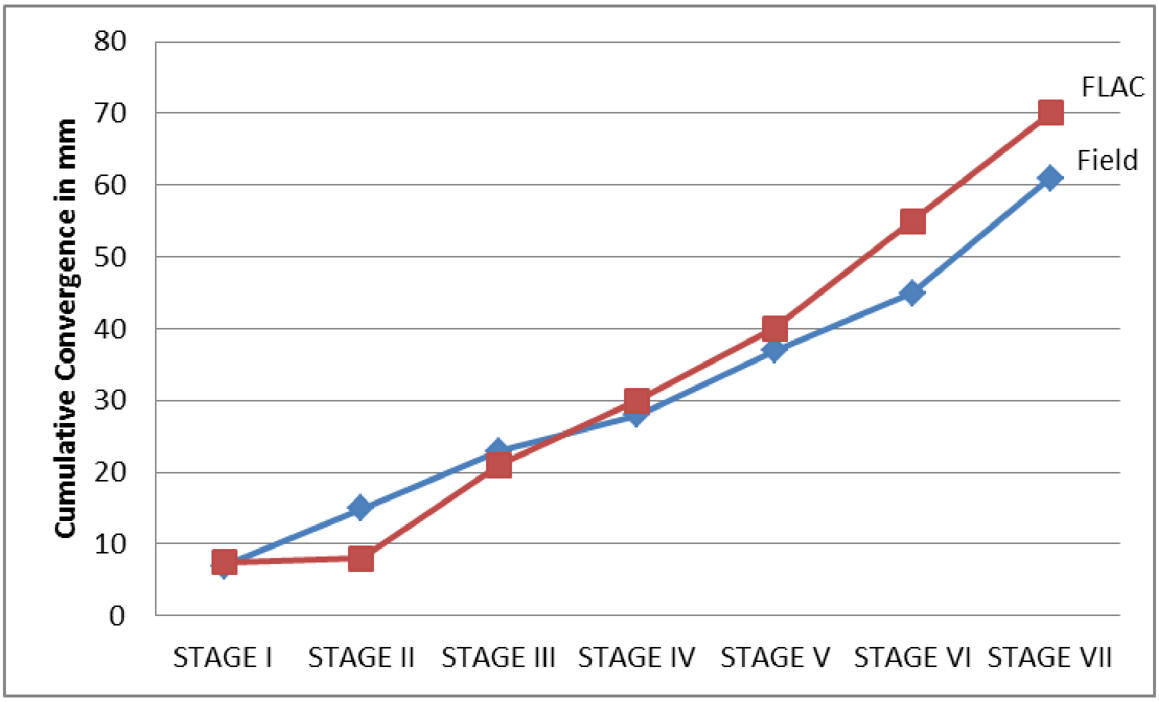

Fig.7.1 Convergence Results: FLAC Results vs. Field Investigation Data

Table 7.1 Comparison of FLAC &. Field Investigation Data

Stage of Extraction	Cum. Convergence in mm (Field)	Cum. Convergence in mm (FLAC)	Percentage Variation (%)
Dev. of galleries	7	7.5	7.1
Dev. of Splits	15	8	46.6
Extraction of Stook 1	23	21	8.7
Extraction of Stook 2	28	30	7.1
Extraction of Stook 3	37	40	8.1
Extraction of Stook 4	45	55	22.2
Extraction of Stook 5	61	70	14.7

The results so obtained by FLAC when compared with that of the Field data, the model prediction is within an approximation of 10% for stages I, III, IV & V whereas for stages VI and VII, it is in 20% approximation. During the development of splits, there was a greater amount of developed insitu stresses which gave rise to consecutive natural falls. As a result, the cumulative convergence measured at the convergence station by the convergence indicator has been high at Stage II. As the area of extraction increased, the area of exposure of roof has also simultaneously increased letting the suspended roof from the goaf edge to act as a suspended cantilever beam. Since the area of exposure increased to a greater extent than which is preferable before goaf settlement, there occurred a continual increase in the cumulative convergence measurements.

CHAPTER 8

CONCLUSIONS

&

SUGGESTIONS

8. CONCLUSIONS & SUGGESTIONS

Based on the study of strata behaviour during extraction of pillars by BG method in 3A panel of GDK 10 Incline, the following conclusions were drawn:

8.1 Conclusions

- Maximum rate of convergence and cumulative convergence recorded in field was about 4mm/day and 61mm respectively, measured at convergence station C-5 in 68 Level.
- Before the occurrence of main fall, rate of convergence showed a continuously increasing trend for five days. This may be considered as warning for impending main fall.
- From the triaxial testing, the major principal stresses of 22, 32 and 41.5 MPa were obtained at confining stresses of 0, 2 and 3 MPa respectively. The plot between the major and minor principal stresses had shown a trend line having an equation, $y = 4.875x + 22.083$ with an R^2 value of 0.9998
- The results obtained from the RocLab software indicated the Cohesion, Friction Angle, UCS and Tensile Strength values to be 1.1MPa, 30.84^0, 1.314 MPa and 0.32MPa respectively.
- Numerical modeling results indicated absolute roof deformation of about 7.5 and 8 mm in the gallery and development splits, and also a maximum cumulative convergence of 70mm at the time of major roof fall respectively.
- The results obtained by FLAC when compared with that of the Field data, the predicted values were within an approximation of 10% for stages I, III, IV & V whereas for stages VI and VII, its in 20% approximation except for that of stage II which showed a comparatively higher cumulative convergence values due to the occurrence of natural fall.

8.2 Suggestions

There is a lot of scope in this study for future work. The study can be carried out more efficiently in the following manner:

- More number of panels can be studied so as to better understand the strata behavior and simulate the conditions in models.

- Instead of two dimensional models, three dimensional modeling can predict strata behavior of the panel more efficiently.

8.3 Limitations of the present work

- Calculation of the insitu stresses acting upon the pillars. Theoretical stress has been considered in this work whereas insitu stresses can be found by hydraulic fracturing method.
- Consideration of all the parameters leads to an erroneous results in numerical modelling. It may be tempting to consider all the parameters into account but it is advisable to consider the minimum and most important parameters only.
- The inability of installing instruments at all points in a panel leads to certain limitations in the study.

CHAPTER 9

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9 REFERENCES

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APPENDIX I

Appendix I - Details of the Panel

At GDK 10 Incline of Adriyala project area SCCL, it is proposed to adopt Blasting Gallery method in panel # 3A of III Seam in Block C. Average depth cover and thickness of the seam in the proposed panel are about 350 m and 11 m, respectively. Geotechnical properties of the strata and observations in the previously worked panels were also considered with particular reference to strata control problems and occurrence of spontaneous heating/fire and associated sealing of some of the previous panels in the mine.

The coal formations of Ramagundam area is of Kamthi and Barakar series. A typical borehole section in the mine area is shown in Fig 10.2. Five workable coal seams occur in the Barakar stage – II, IIIB, IIIA, III and IV seams. The top most seam I is being worked by GDK 10A Incline. Seams IA and IIIB are inconsistent, and therefore could not be worked. The lower most seams IIIA, III and IV are being worked by GDK 10 Incline. Two major faults are running through the property. One of that has up throw of about 53 m while another is a down throw fault with a throw of about 24 m. The mine property is divided into three blocks: Block A, Block B and Block C, demarcated by the fault running across the property.

Geomining Conditions of the Panel – 3A

The coal measure formations observed in borehole # 441 within GDK-10 Incline area are shown in Fig 10.2. Thickness of #3 seam is about 11 m with an average gradient of 1 in 5.5. The strata overlying the coal seam are composed of white sandstone with carbonaceous clay bands. Coal face mechanization in the panel consists of jumbo drills and remote controlled Load Haul Dumpers (LHD) loading on to chain conveyors in the levels.

Table T1: Details of working BG Panel No.3Aof No.3 Seam, Block-C

Incubation period	9 months
Overlying Seam	1 Seam goaf by 10A Longwall
Underlying Seam	4Seam virgin
Thickness of seam	11 m
Width of the development gallery	4.2 m
Height of the development gallery	3.0 m
Size of the Panel	150m x 120m
Area of the Panel	16000 m^2
No. of Pillars	6

No. of Rooms		9
Depth	Min:	323 m
	Max:	352 m
Average size of pillars		60 X 50
Gradient of the seam		1 in 5.5
Boundary		
North		3 seam virgin.
East		3 seam virgin.
West		3 seam 2A panel sealed off area.
South		3 seam bottom section B&P developed area, standing on pillars
Total Coal in the Panel		283000 T

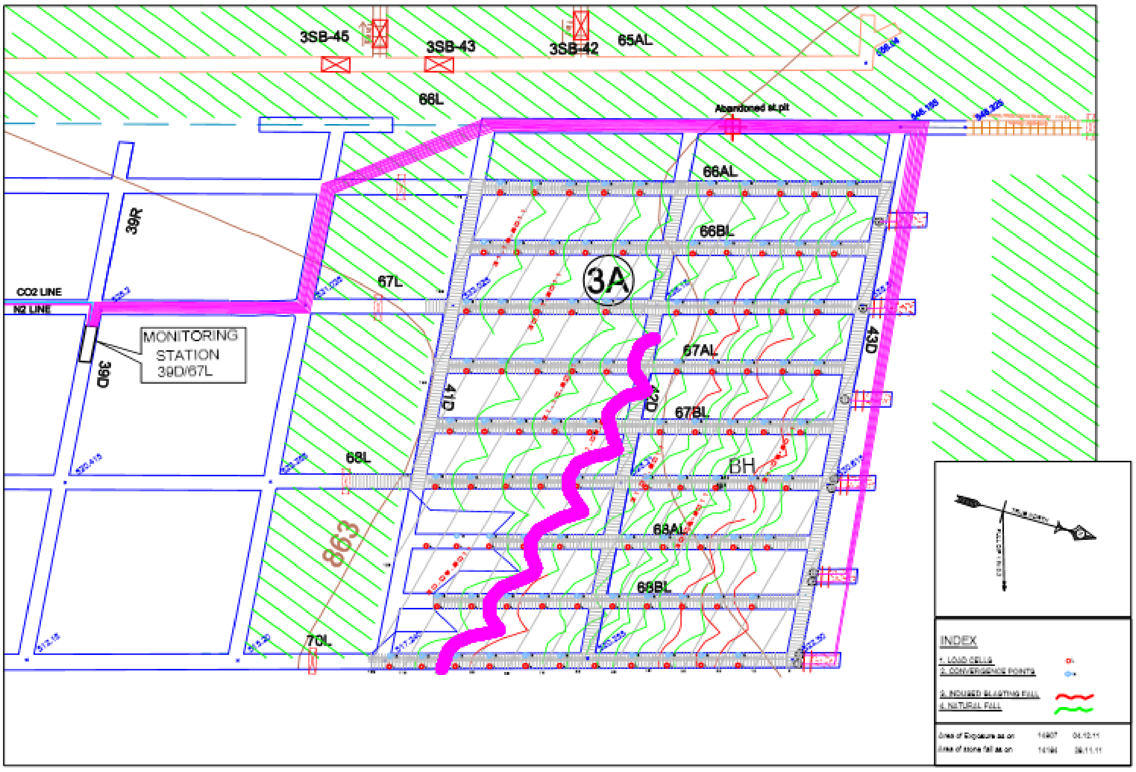

Fig A1: BG Panel #3A, GDK 10 Incline, SCCL Layout

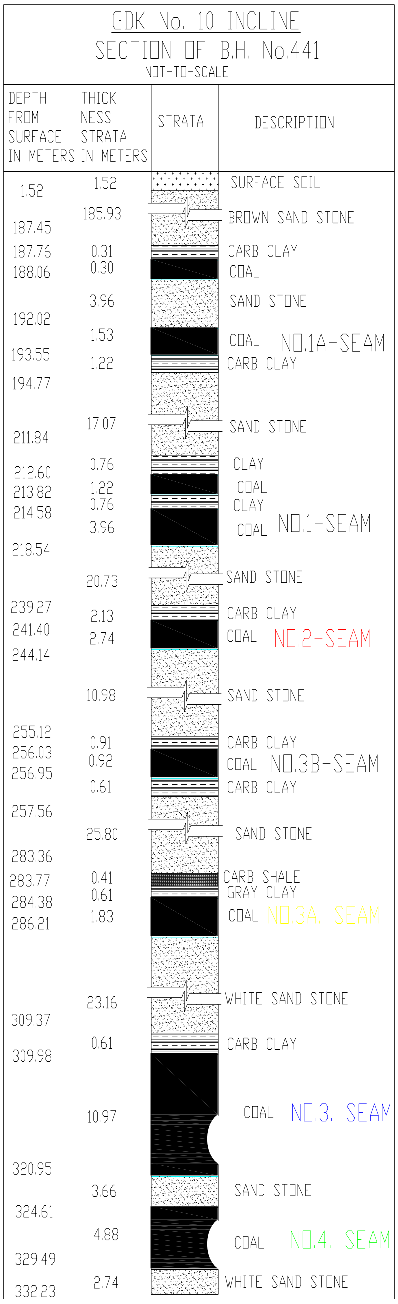

Fig: A2: Borehole Section Of BH No. 441 at GDK 10 Inc, SCCL

10.1.2 Support system in the BG-3A panel

The support system in the district consists of I-section MS cross girders of 200 x 200 mm, set on 40 ton hydraulic props at each end . In each row there are two props and a girder, with a row spacing of 1.0 m. Additional supports including chocks and props are being provided wherever required. The split galleries are supported with 1.8 m long roof bolts with 1m spacing and row is 1.2m apart. Advance supports are installed up to 40m in all the rooms. Junctions are supported by two sets of skin to skin MS girders of 150mm x 150mm and supported by two No. of 40T hydraulic props on each side. In addition to the above cable bolting was done at 1.5m interval in grid pattern anchored up to a length of 1.0 m above the coal seam into sand stone roof. Corners and Sides supporting is being done with 1.5 m length bolts with 1m grid pattern whenever required.

10.1.3 Additional support

Entries to the central dip rise galleries of panel immediately out bye of the goaf edges was kept supported by cogs set at an interval of 0.25 m. Before commencing drilling by jumbo drills, the goaf edge of gallery was kept supported by a row of props erected at interval of 0.5 m. Wire meshing is fixed where ever the height of gallery made was more than 4m.

10.1.4 Measures against strata control problems

Density of supports was increased by decreasing the span between girders. Grouting of 4 rows of 1.8 m. roof bolts was done at 1 m. grid. Side bolting is done in all the galleries. Regular induced blasting is being carried out up to 1.5 m. in the Sand stone roof. Frequent re-setting of hydraulic props is being done due to probability of disturbance of vertical supports by moving machinery

10.1.5 Measures adopted to prevent strata control problems

Diagonal line of extraction was ensured to avoid strata control problems. Additional OC props were erected up to 5m in each room. All the OC props within 10m distance were retightened before drawing goaf edge girders for ring blasting. Regular induced caving was ensured by blasting the stone. Leaving of stooks inside the goaf was minimized to prevent transfer of strata pressures on working area. Faster and regular rate of retreat was ensured with working on holidays.

Appendix II – Numerical Modelling Code

BG 3A PANEL, GDK 10 Incline, SCCL

```
TITLE
STRATA BEHAVIOUR ANALYSIS IN BG3A SEAM - GDK 10 INCLINE
*PROGRAM DEVELOPED BY B.N.V. SIVA PRASAD
* SEAM THICKNESS=11M, PILLAR SIZE=50M, DEPTH=350M
* GALLERY SIZE=4.2M X 3M
GR 147 44
M M
*
* FLOOR OF THE SEAM NO 3
GEN 0,0 0,150 60,150 60,0                              R .8 .8 I 1 12 J 1 15
GEN 60,0 60,150 64.2,150 64.2,0                        R 1 .8 I 12 16 J 1 15
GEN 64.2,0 64.2,150 114.2,150 114.2,0                  R 1 .8 I 16 52 J 1 15
GEN 114.2,0 114.2,150 118.4,150 118.4,0                R 1 .8 I 52 56 J 1 15
GEN 118.4,0 118.4,150 168.4.2,150 168.4,0              R 1 .8 I 56 92 J 1 15
GEN 168.4,0 168.4,150 172.6,150 172.6,0                R 1 .8 I 92 96 J 1 15
GEN 172.6,0 172.6,150 222.6,150 222.6,0                R 1 .8 I 96 132 J 1 15
GEN 222.6,0 222.6,150 226.8,150 226.8,0                R 1 .8 I 132 136  J 1 15
GEN 226.8,0 226.8,150 286.8,150 286.8,0                R 1.2 .8 I 136 148 J 1 15
*
*COAL SEAM -11M THICK
GEN 0,150 0,161 60,161 60,150                          R .8 1 I 1 12 J 15 22
GEN 60,150 60,161 64.2,161 64.2,150                    R 1 1 I 12 16 J 15 22
GEN 64.2,150 64.2,161 114.2,161 114.2,150              R 1 1 I 16 52 J 15 22
GEN 114.2,150 114.2,161 118.4,161 118.4,150            R 1 1 I 52 56 J 15 22
GEN 118.4,150 118.4,161 168.4.2,161 168.4,150          R 1 1 I 56 92 J 15 22
GEN 168.4,150 168.4,161 172.6,161 172.6,150            R 1 1 I 92 96 J 15 22
GEN 172.6,150 172.6,161 222.6,161 222.6,150            R 1 1 I 96 132 J 15 22
GEN 222.6,150 222.6,161 226.8,161 226.8,150            R 1 1 I 132 136 J 15 22
GEN 226.8,150 226.8,161 286.8,161 286.8,150            R 1.2 1 I 136 148 J 15 22
*
* SANDSTONE ROOF
GEN 0,161 0,511 60,511 60,161                          R .8 1.2 I 1 12 J 22 45
```

```
GEN 60,161 60,511 64.2,511 64.2,161                  R 1 1.2 I 12 16 J 22 45
GEN 64.2,161 64.2,511 114.2,511 114.2,161            R 1 1.2 I 16 52 J 22 45
GEN 114.2,161 114.2,511 118.4,511 118.4,161          R 1 1.2 I 52 56 J 22 45
GEN 118.4,161 118.4,511 168.4.2,511 168.4,161        R 1 1.2 I 56 92 J 22 45
GEN 168.4,161 168.4,511 172.6,511 172.6,161          R 1 1.2 I 92 96 J 22 45
GEN 172.6,161 172.6,511 222.6,511 222.6,161          R 1 1.2 I 96 132 J 22 45
GEN 222.6,161 222.6,511 226.8,511 226.8,161          R 1 1.2 I 132 136 J 22 45
GEN 226.8,161 226.8,511 286.8,511 286.8,161          R 1.2 1.2 I 136 148 J 22 45
PROP S=4.E9 B=6.67E9 D=2300 T=9.E6 C= 12.E6 FRIC=45 I 1 147 J 1 14
PROP S=4.E9 B=6.67E9 D=2300 T=9.E6 C=12.E6 FRIC=45 I 1 147 J 22 44
PROP S=2.2E9 B=3.67E9 D=1427 T=0.32E6 C=1.1E6 FRIC=30.84 I 1 147 J 15 21
SET GRA 9.81
SET LARGE
FIX X I 1
FIX X J 1
FIX X I 148
FIX Y J 1
INI SYY -11.5E6 VAR 0 11.5E6
INI SXX -3.77E6 VAR 0 3.77E6
HIS NSTEP 10
HIS XDIS I 148 J 17
HIS YDIS I 148 J 17
*DEVELOPMENT GALLERIES 4.2M X 3M
HIS UNBAL I 1 J 1
********************OPENING OF GALLERY 1***********
MOD NULL I 12 15 J 15 16
********************OPENING OF GALLERY 2***********
MOD NULL I 52 55 J 15 16
********************OPENING OF GALLERY 3***********
MOD NULL I 92 95 J 15 16
********************OPENING OF GALLERY 4***********
MOD NULL I 132 135 J 15 16
S = 100
SAVE BG3ADEV.SAV
```

****

******SPLIT GALLERIES 4.2M X 3M

********************OPENING OF SPLIT 1**********

MOD NULL I 119 121 J 15 16

********************OPENING OF SPLIT 2**********

MOD NULL I 106 108 J 15 16

S = 100

SAVE BG3ASP2.SAV

********************OPENING OF SPLIT 3**********

MOD NULL I 79 81 J 15 16

********************OPENING OF SPLIT 4**********

MOD NULL I 66 68 J 15 16

********************EXCAVATION FROM STOOK 1

MOD NULL I 126 135 J 15 22

MOD NULL I 114 126 J 15 22

S = 100

SAVE BG3ASP4ST1.SAV

********************OPENING OF SPLIT 5**********

MOD NULL I 39 41 J 15 16

********************OPENING OF SPLIT 6**********

MOD NULL I 26 28 J 15 16

****

********************EXCAVATION FROM STOOK 2

MOD NULL I 101 113 J 15 22

S=100

SAVE BG3ASP6ST2.SAV

********************EXCAVATION FROM STOOK 3

MOD NULL I 87 100 J 15 22

S=100

SAVE BG3ASP6ST3.SAV

********************EXCAVATION FROM STOOK 4

MOD NULL I 74 86 J 15 22

S=100

SAVE BG3ASP6ST4.SAV

```
********************EXCAVATION FROM STOOK 5
MOD NULL I 61 73 J 15 22
S=100
SAVE BG3ASP6ST5.SAV
********************EXCAVATION FROM STOOK 6
MOD NULL I 47 60 J 15 22
S=100
SAVE BG3ASP6ST6.SAV
********************EXCAVATION FROM STOOK 7
MOD NULL I 34 46 J 15 22
S=100
SAVE BG3ASP6ST7.SAV
********************EXCAVATION FROM STOOK 8
MOD NULL I 21 33 J 15 22
S=100
SAVE BG3ASP6ST8.SAV
RET
```